The energy analysis of burner reactor power systems

Thesis

THE ENERGY ANALYSIS OF BURNER REACTOR POWER SYSTEMS

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by Nigel David Mortimer, BSc.,

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ABSTRACT

Currently most commercial nuclear power stations are based on thermal reactor designs called burner reactors which are net consumers of fissile material. These power stations form one part of a larger system that generates electricity from uranium. However, in addition to producing energy, such systems also consume energy, in the form of various fuels, during construction and operation. This thesis describes the use of energy analysis to determine the total energy required by these systems.

A number of factors are shown to influence energy consumption and, in particular, the effect of extracting uranium from different sources is studied in detail. For ores, an important inverse relationship between energy use and ore grade is investigated and quantified. The physical limit at which the energy input to the system is equal to its output is shown to correspond to an average grade of 15 parts per million of "triuranium octoxide". Analysis of proposals for extracting uranium from seawater indicates that the only schemes giving a positive energy balance are costly ($500/lb U_3O_8) and limited to low production rates.

The effects of feedback within fuel systems are analysed and the results are used to formulate an economic model in which nuclear electricity prices determine uranium ore costs as well as vice versa. The model demonstrates that, with present techniques, the average economic limit to ore grade is 50 ppm U_3O_8 with subsequent resources, on current assessment, of only 10^7 tonnes U_3O_8. This contradicts most traditional studies which, by assuming fixed, non-dependent fuel costs, suggest an ore grade limit of less than 4 ppm U_3O_8 and economically recoverable resources in excess of 10^{10} tonnes U_3O_8.

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1 GENERAL INTRODUCTION

1.1 Energy analysis

Although energy can be simply defined as the capacity for doing work, it appears in many different guises and originates from various sources. Energy analysis is the study of the use of energy in the production of goods and the provision of services and hence the most important energy sources examined here are those called fuels. These sources of energy are consumed by machines and appliances to provide heat, light, motive power, etc., and the current list of common fuels includes coal, petroleum products, natural gas and electricity. Feedstocks, which are fuels used as sources of raw materials rather than sources of energy, are also included within the scope of energy analysis, whilst energy sources such as labour and food are excluded.

Fuels provide the energy employed in processes to effect changes in materials. To transform a naturally-occurring material, or resource, into a final product, individual processes are frequently combined into groups known as production systems. The total industrial system which supplies all the goods and services required by society is a collection of these production systems. Since energy in the form of fuels is a fundamental ingredient of industrial operations, energy analysis is an important means of studying certain aspects of this complete system.

Chemical and production engineers sometimes investigate the use of energy in specific processes to discover ways of minimising fuel consumption and production costs. This can involve the

comparison of actual fuel consumption with the theoretical or practical minimum amount of energy required by processes. Such analysis can reveal the immediate effects and direct significance of particular fuel conservation measures.

To examine the wider implications of fuel conservation and, on a national scale, fuel policy, it is necessary to look beyond individual processes and investigate the total use of energy in the entire industrial system. This arises because processes use goods and services as well as fuels and these goods and services also consume fuel during their manufacture. Hence the total amount of energy required to produce an item equals not only the amount used directly in main production operations but also that consumed by all preceding ancillary processes. Consequently energy analysis is concerned with the investigation of both direct and indirect energy consumption.

Of all the processes incorporated into the industrial system, one of the most interesting groups to study by energy analysis are the fuel supply systems. These manufacture fuels from natural sources of energy known as energy resources. Energy analysis can be used to examine ways in which different fuel supply systems utilise energy resources and the particular topic investigated in this thesis is the operation of nuclear fission power systems.

1.2 Area of study

A nuclear fission power system is a collection of processes that produce fuel from certain energy resources by means of nuclear fission reactions. These reactions can occur when nuclei absorb neutrons and split apart. As a result of a small

loss of total mass, energy is released and during ensuing nuclear rearrangements neutrons are emitted which may initiate further fission reactions. Consequently under favourable conditions such reactions can be self-sustaining sources of energy. Nuclei capable of undergoing these important reactions are called fissile.

The ability of a nucleus to fission is determined by the number of protons and neutrons it contains. Nuclei of the same element which have different numbers of these particles are known as isotopes. Important fissile isotopes include those of uranium which contain a total of 233 and 235 protons and neutrons, U-233 and U-235 respectively, and the plutonium isotope Pu-239. Although only the isotope U-235 occurs naturally, both U-233 and Pu-239 can be produced artificially from natural isotopes which are subsequently referred to as fertile.

Fissile U-233 can be obtained from the fertile thorium isotope Th-232 by the absorption of neutrons and the fertile uranium isotope U-238 can be converted to fissile Pu-239 by a similar process. The importance of these conversion reactions is that both U-238 and Th-232 are more naturally abundant than U-235. The amount of U-235 in natural uranium is only 0.71% by mass compared to 99.28% U-238, and natural thorium consists entirely of Th-232. Hence the relative amount of energy resources available for any given nuclear fission power system largely depends on the specific nuclear reactions utilised.

The equipment used to provide the necessary fission and conversion reaction conditions is called a nuclear reactor and there are three general types. These utilise different

reactions and are known as burner, converter and breeder reactors. Since both burner and converter reactors rely on reactions that involve slow-moving, or thermal, neutrons, such designs are often referred to as thermal reactors. Breeder reactors generally use reactions requiring neutrons that travel at much higher speeds and are consequently called fast reactors.

The main source of energy in a burner reactor is the fission reaction between thermal neutrons and fissile isotopes of uranium. Despite occasional conversion reactions which produce Pu-239, a net loss of fissile isotopes occurs during the operation of such designs and this accounts for their title of 'burners'.

Reactors which rely mainly on conversion reactions for fissile isotopes are known as 'converters'. One particular example of this type operates on the transformation of Th-232 by neutrons to U-233 which then provides energy by fission.

Breeder reactors primarily depend on the fission of Pu-239 by fast neutrons as a source of energy. Surplus Pu-239 is produced during operation by the conversion of U-238 by fast neutrons and the term 'breeder' originates from the ability of such reactors to create a net gain in this particular fissile isotope.

Apart from the reactor physics, certain other features of nuclear power systems reflect the differences between these basic reactor types. This is particularly apparent in that part of the system known as the nuclear fuel cycle which is the group of processes that supply suitable fissile and fertile

isotope material for reactor use. Most of the fuel cycle of burner reactor power systems is devoted to the production of uranium from resources. The processing of thorium would be a fundamental component of the converter fuel cycle and the treatment of plutonium would be an important part of the breeder fuel cycle.

In addition to the fuel cycle and reactor, nuclear power systems also contain means of producing and distributing saleable fuel from the energy released by fission reactions. Despite differences in basic reactor types, these parts of the system are usually quite similar. Heat from nuclear reactions is converted to electricity by heat-exchange equipment and turbo-electric machinery situated, along with the reactor, in a group of buildings known as the nuclear power station. This is in turn connected to transmission and distribution equipment which delivers electricity to consumers.

Although a nuclear fission power system is composed of a fuel cycle, power station and electricity transmission network, the basic characteristics of the reactor alone can significantly influence its operation. In particular, the ability of converter and breeder reactors to produce fissile from non-fissile isotopes means that systems based on these designs can utilise nuclear energy resources more effectively than burner reactor systems. Both converter and breeder reactors, however, are still in the early stages of development and final details of design and operation have yet to be determined. In contrast, commercial burner reactors of one sort or another are already in use and are expected to play a significant role in the growth of world fuel supply. As a consequence, this study

is specifically concerned with the analysis of nuclear fission power systems which incorporate common burner reactor designs.

1.3 The project

The purpose of the project described in this thesis is to evaluate the total amount of energy required to produce electricity from burner reactor power systems. This involves detailed examination of all stages in the fuel cycle, power station construction and operation, parts of the electricity transmission and distribution network, and certain processes that supply the system with important goods and services. The significance of factors which influence the use of energy by the system is also investigated.

In addition to the measurement of energy use, practical applications of results are demonstrated. Areas of potential fuel conservation are identified and savings are compared with total consumption. The role of fundamental variables in determining the amount of energy required by the system is translated into economic terms and this is related to the availability of nuclear energy resources. The resulting prospects for the burner reactor are assessed and some of the consequences for the nuclear power industry are discussed.

2 METHODOLOGY

2.1 Basis of analysis

The basis of energy analysis is a consistent set of definitions and conventions used to evaluate and present results. This section describes all the essential details of the particular basis adopted in this study and enables results to be estimated correctly and interpreted unambiguously.

Energy analysis is concerned with the measurement of the use of energy. Energy is the capacity for doing work on either the microscopic or macroscopic level and consequently occurs in many forms, ranging from the random motion of molecules called heat to the popular image of work in the mechanical sense as the uniform motion of matter.

In thermodynamic terms energy analysis can be described as the study of free energy changes. Free energy is a thermodynamic potential which indicates the amount of energy liberated or absorbed in a reaction. Free energy changes can be deduced theoretically assuming all processes are reversible. However, since real reactions proceed at finite rates and involve frictional and similar forces, it is generally more practical to discuss actual free energy changes associated with irreversible processes.

Although the principles of energy analysis can be formally described in terms of free energy changes, in practice it is more convenient to measure enthalpy changes. Enthalpy is a thermodynamic property representing the heat content of a substance. Despite distinctions between free energy and enthalpy, the numerical difference for the energy released by

the combustion of most common fuels is generally regarded as unimportant in energy analysis.

The enthalpy, or quantity of heat released during the total combustion of a given amount of fuel, feedstock or energy resource, is commonly referred to as the energy content, or calorific value. There can be significant differences between the energy content of an energy resource and that of its subsequent fuel or feedstock due to thermodynamic and technical factors involved in conversion processes. For example, the energy potentially available from the fission of all the U-235 nuclei in one gramme of natural uranium is approximately five times the amount of electrical energy that it can produce in a burner reactor power station.

The amount of energy released by, or associated with the use of, fuels and feedstocks in a process is found by multiplying fuel and feedstock consumption by the appropriate values of energy content. This is referred to as the direct energy input to the process and, where fuels are consumed in particular, it consists of energy which performs specific useful functions as well as superfluous heat or energy losses. The ratio of useful energy to the direct energy input for any operation equals the energy efficiency and this parameter varies according to such factors as the sort of equipment used, the way it is controlled and the type of fuel consumed.

The consumption of energy in the manufacture of the fuels, feedstocks, goods and services used by a process introduces an indirect energy input to that process. Hence the total amount of energy required to make one unit of output by the process equals the sum of both direct and indirect energy inputs; this

being known as the energy requirement (e.r.) if the output is measured in physical terms (eg. energy per unit mass), or the energy intensity (e.i.) if measured in financial terms (eg. energy per pound sterling). Various general inputs that can influence these results are shown in figure 2.1 which also illustrates the flowchart symbols adopted here.

Since all processes that contribute to the production of an item must be taken into account when evaluating its energy requirement or energy intensity, such results indicate the total amount of energy, in all forms, participating in its manufacture. Naturally-occurring energy resources initially supply all this energy and consequently results incorporate energy content contributions of these basic inputs. Hence results of energy analysis can be used to determine the effects of industrial activity on resources and total heat release.

As mentioned earlier, the energy content of resource and resulting fuel can be significantly different and, depending on the way results are to be used, the energy requirement of a fuel can be defined to either include or exclude the energy content of the resource from which it is made. Therefore the analysis of fuel supply systems can produce two basic results; the gross energy requirement and the net energy requirement.

The gross energy requirement (g.e.r.) of a fuel is the sum of all the direct and indirect energy used during manufacture plus the energy content of the original resource. The net energy requirement (n.e.r.) excludes this last term.

The general principles used to evaluate and interpret these two results can be explained by referring to figures 2.2, 2.3 and 2.4 which show groups of processes, conveniently indicated as

Figure 2.1 : Energy inputs to a single process stage

(cf. International Federation of Institutes for Advanced Studies, 1975)

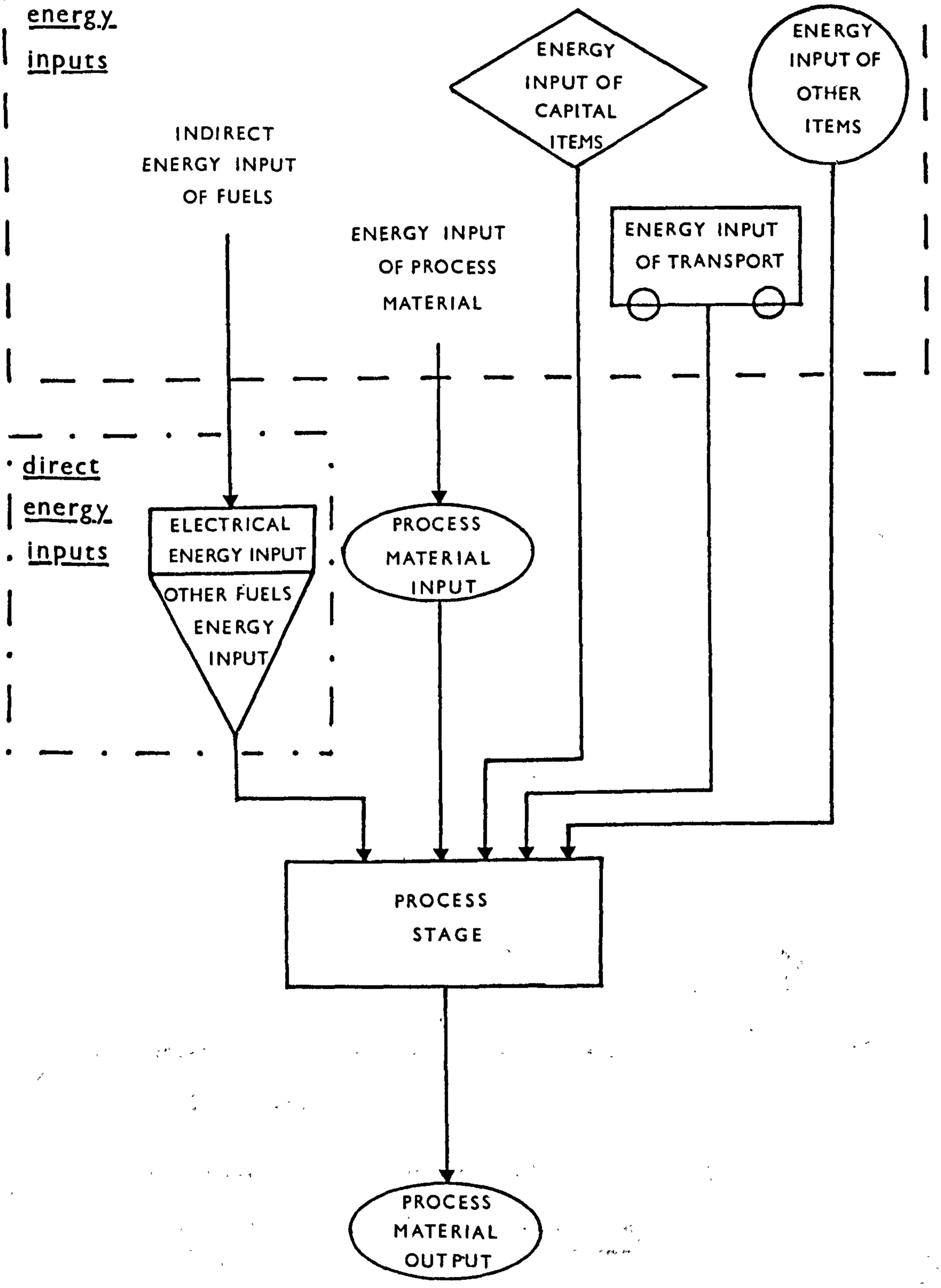
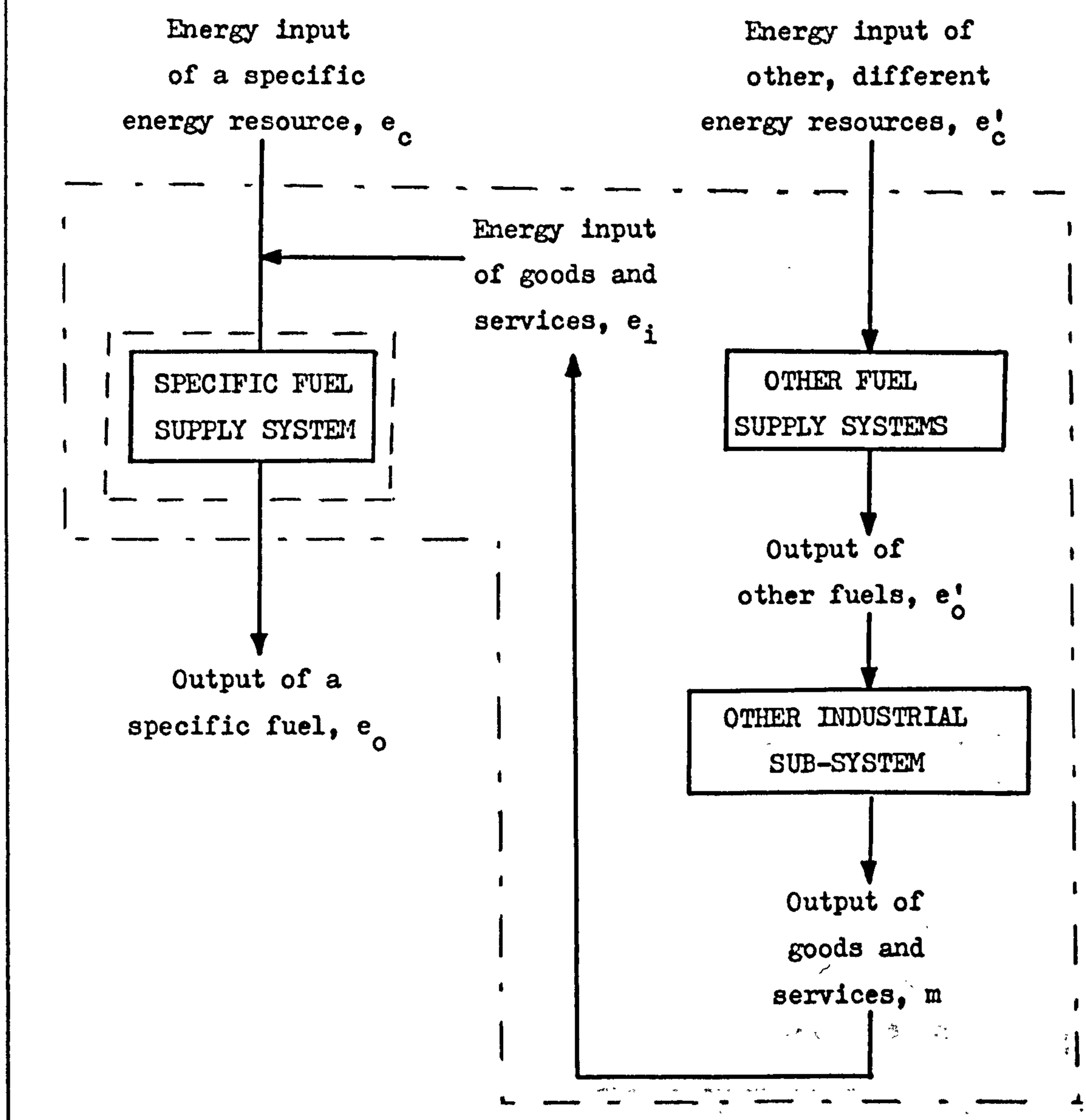


Figure 2.2 : Evaluation of the gross energy requirement of a fuel.



Energy requirement of goods and services, $E_M = \frac{e'_o}{m} = \frac{e'_c}{m}$

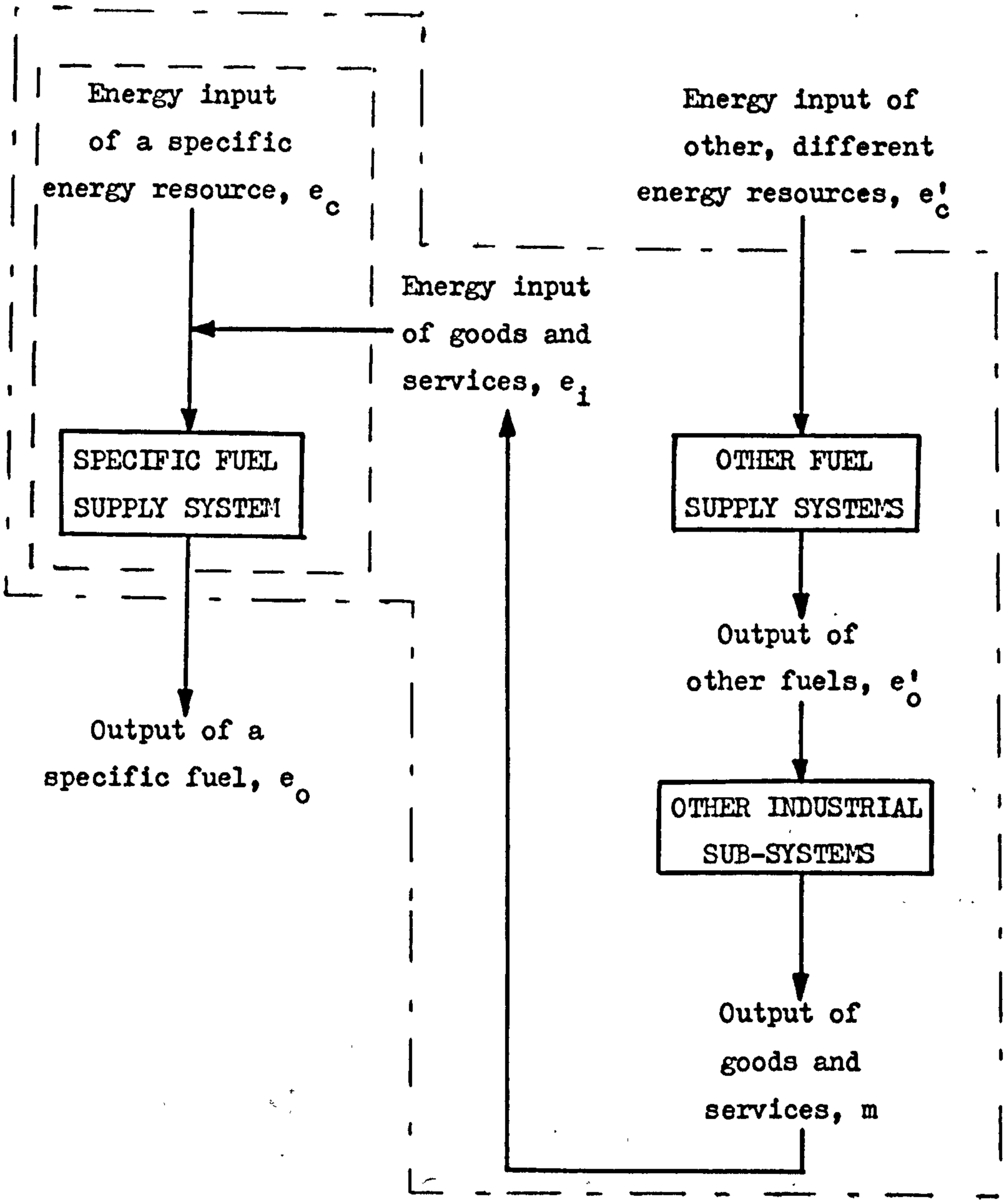
Hence, energy input of goods and services to the specific fuel supply system, $e_i = E_M \times m = e'_c$

$\therefore$ Gross energy requirement of the specific fuel, E_G

$$= \frac{e_c + e_i}{e_o} = \frac{e_c + E_M \times m}{e_o} = \frac{e_c + e'_c}{e_o}$$

NOTE : $e_c + e'_c$ = energy input of ALL energy resources

Figure 2.3 : Evaluation of the net energy requirement of a fuel - Case 1.

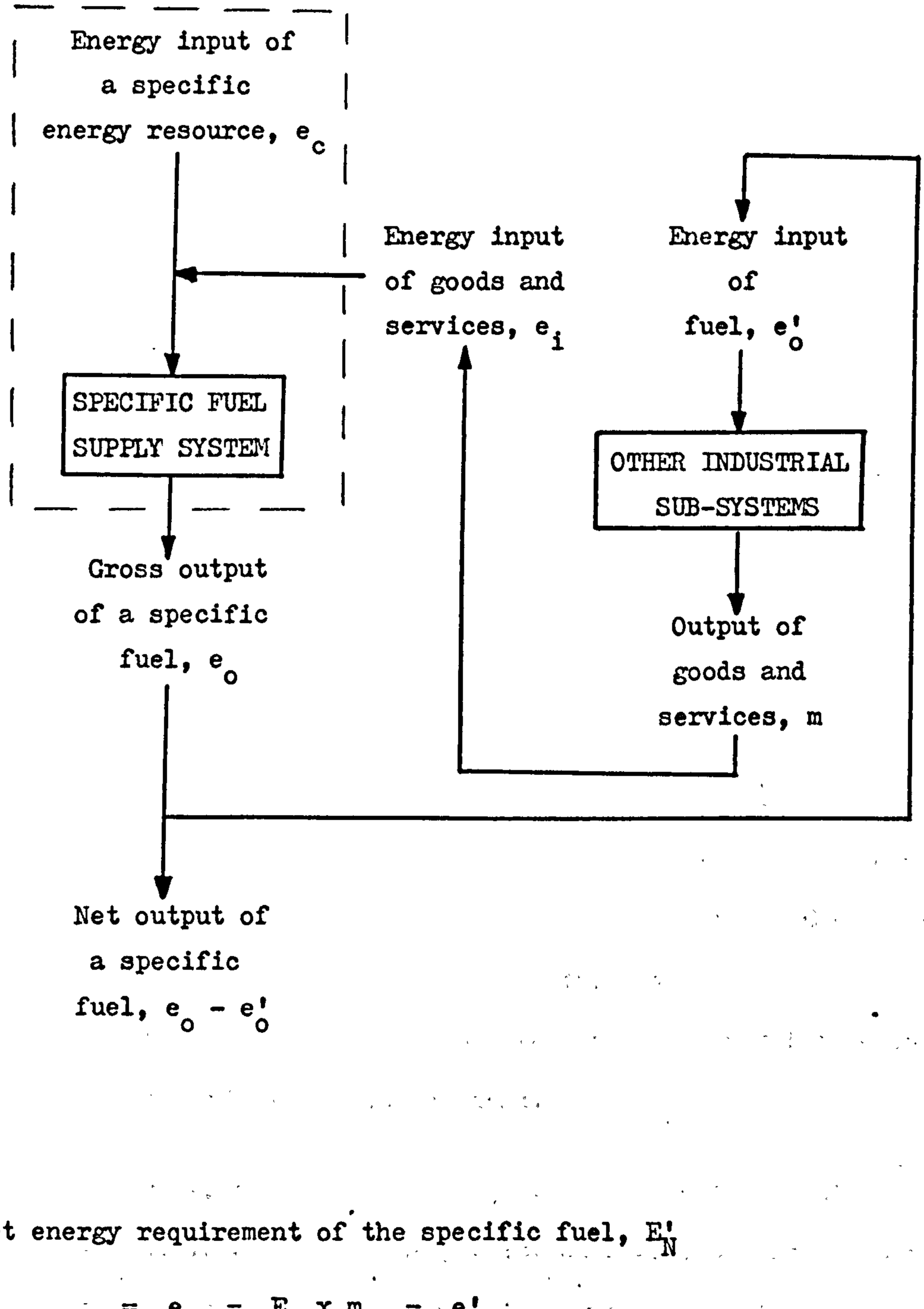


Net energy requirement of the specific fuel, E_N

$$= \frac{e_i}{e_o} = \frac{E_M \times m}{e_o} = \frac{e'_c}{e_o}$$

NOTE : e'_c = energy input of OTHER energy resources

Figure 2.4 : Evaluation of the net energy requirement of a fuel - Case 2.



Net energy requirement of the specific fuel, E'_N

$$= \frac{e_i}{e_o} = \frac{E_M \times m}{e_o} = \frac{e'_o}{e_o}$$

NOTE : e'_o = energy input of fuel used within the two inter-connected systems = FUEL FEEDBACK

sub-systems, combining together to produce a particular fuel. The basic components of these figures consist of fuel supply systems and an industrial sub-system which provides all necessary goods and services. A diagram representing the production of coal, for example, would contain enterprises which make mining equipment, lubricating oil, electricity, etc., as well as the actual coal mine itself.

Figures 2.2 and 2.3 show a simplified arrangement of processes which produce one particular fuel from a specific fuel supply system that uses products manufactured with fuels from other, quite different supply systems. Such an arrangement would correspond to the case of a newly-introduced fuel industry which relies entirely on the support of the existing industrial infrastructure.

Figure 2.2 indicates how the gross energy requirement of a fuel would be calculated in this situation. By definition this result represents the total energy input of all energy resources involved in the production of the fuel. Therefore the gross energy requirement can be used to study total resource utilisation, environmental heat release and the relative energy efficiency of fuel supply systems.

In contrast, the net energy requirement of a fuel excludes the energy contribution of the particular resource from which it originates and hence this result expresses the amount of energy released by the specific use of fuels in the entire system. As figure 2.3 shows, such a result indicates how the use of one particular energy resource affects all others associated with its exploitation.

Figures 2.2 and 2.3 represent a situation in which one sub-system is wholly dependent on others for manufactured products. The alternative case is illustrated in figure 2.4 which shows a fuel industry producing all the fuel for processes that in turn supply it with all necessary operating requirements. Such a self-sustaining arrangement could occur in a well-established industrial system and therefore figure 2.4 indicates what happens when one fuel totally dominates energy supply. The net energy requirement of a fuel produced under these conditions reflects the ability of the system to support its own fuel needs. Evaluation of the net energy requirement of fuel produced by such a system involves measuring the fuel feedback, or portion of fuel output that the system itself consumes. As a consequence, this result can be used to assess the absolute fuel efficiency of a fuel supply system.

The energy requirement of any product is generally deduced by multiplying total fuel consumption by appropriate energy requirement values. Where the product is a fuel, however, the calculation can be complicated by the fact that the fuel supply system can provide all the fuel needed by its associated industrial sub-system. For example, a nuclear power system can rely on fuels produced from coal, oil and natural gas (cf. figures 2.2 and 2.3), or it can consume part of its own electrical output (cf. figure 2.4).

To estimate the energy requirement of electricity generated by the nuclear power system in the former case, each type of fuel input is simply multiplied by appropriate gross energy requirements obtained from previous analyses. Figure 2.3

provides the basic equation to evaluate the net energy requirement;-

$$\text{Net energy requirement} = \frac{\sum_i (\text{total fuel consumption})_i \times (\text{fuel g.e.r.})_i}{\text{gross electricity output}}$$

where i refers to all fuels including electricity

In the intermediate situation where the nuclear power system provides all its own electricity and other systems supply all the remaining fuels, the equation becomes;-

$$\text{Net energy requirement} = \frac{\sum_j (\text{total fuel consumption})_j \times (\text{fuel g.e.r.})_j}{(\text{gross electricity output}) - (\text{total electricity consumption})}$$

where j refers to all fuels excluding electricity

The energy requirement of electricity from a system which provides all its own fuel is obtained by referring to the equations in figure 2.4 which can be summarised as;-

$$\text{Net energy requirement} = \frac{\text{total electricity consumption}}{\text{gross electricity output}}$$

$$= \frac{\text{electricity feedback}}{\text{gross electricity output}}$$

These changes in the assumptions about the source of electricity used in the nuclear power system can consequently be accommodated in the evaluation of energy requirements by expressing calculations separately in terms of the direct input of electrical energy and the total energy input of all

other fuels. This is an important feature of this particular analysis and it is discussed further in the following section on units.

2.2 Units

At the present time a wide range of units are being used in energy analysis. Common units of energy include the kilo-watt hour (kWh), the British thermal unit (Btu), the metric ton of coal and oil equivalent (t.c.e. and t.o.e., respectively) and the mega-joule (MJ). A diverse set of units are also used to measure other quantities such as mass, volume, etc. However, an attempt has been made to unify the units of measurement in energy analysis on the basis of the Systeme Internationale (SI) d'Unites (International Federation of Institutes for Advanced Studies, 1975). SI units are adopted here with the addition of convenient and derived units for energy, mega-joules (MJ: 10^6 joules); mass, tonne (te: 10^3 kilo-grammes); and capacity, litres (l: 10^{-3} metres).

In this particular study the presentation of results incorporates a distinction between electricity and other fuels such as coal, oil, gas, etc. This enables nuclear fission power systems to be analysed using different assumptions about the specific source of the electricity they consume. The direct energy input of electricity is consequently indicated by the notation 'mega-joules electrical', MJ(e), and the total energy input of all other fuels by 'mega-joules thermal', MJ(t). Hence, the energy requirement of a kilo-gramme of stainless steel is expressed as;-

$$18\ \text{MJ(e)} + 36\ \text{MJ(t)}$$

This would mean that the total amount of energy extracted from resources during steel production would be 55 MJ(t) if electricity could be converted from fuels with 100% efficiency or 108 MJ(t) if electricity were provided by fossil fuel-fired plant operating at an average efficiency of 25%.

2.3 Methods

Process analysis and statistical analysis are two general methods currently used in energy analysis to obtain results. Process analysis is a means of producing individual results by investigating specific data from original sources. Statistical analysis is a way of obtaining a wide range of results by examining aggregated data from statistical sources. Although these methods are different, they can be used to complement each other.

Process analysis produces specific, accurate results by the detailed study of individual processes. Although this method basically consists of investigating processes directly involved in the manufacture of given products, to obtain energy requirement results it is also necessary to examine ancillary processes. Even the simplest process can rely on many ancillary processes and the resulting expansion of the analysis to include all subsequent contributions to the energy requirement can be lengthy and complicated. However, since the influence of successive ancillary processes often becomes decreasingly significant, less accurate information is generally used to deduce such indirect energy inputs.

Statistical analysis provides comprehensive, approximate results by the systematic study of tabulated and summarised information. Consequently the results of statistical analysis are frequently used in conjunction with the evaluation of indirect energy inputs for process analysis. The advantage of using statistical data which describe substantial parts of the industrial system is that the interrelation of processes can be easily assessed and incorporated into results. However, the subsequent aggregation of information and the use of financial rather than physical accounting are some of the disadvantages associated with this type of analysis.

The main method used here to investigate nuclear power systems is process analysis. However, both process and statistical analysis are used to deduce the energy requirements of important goods and services needed by these systems. This fundamental information is contained in a data base which was compiled during the early stages of this research.

3 DATA BASE

3.1 Results of statistical analysis

Data bases containing extensive energy analysis information have been compiled from national statistics by a number of researchers (Casper, Chapman and Mortimer, 1974; Herendeen and Bullard, 1974; Wright, 1974). The general methods that were used can be divided into two categories; input-output analysis, which consists of examining economic matrices, and sector analysis, which involves investigating statistical summary tables. The statistical results used here were obtained by sector analysis of the "Report on the Census of Production: UK 1968" (Casper et al, 1974).

Census of Production statistics contain information on the use of fuels, materials and other items by numerous sectors of the UK industrial system in particular years. These statistics are aggregated and those of individual sectors frequently refer to a wide range of organisations engaged in the manufacture of similar, though often non-identical, products. Additionally most of the basic information is described in financial rather than physical terms and consequently analysis mainly produces energy intensity results.

The initial step in the analysis of the Census of Production statistics consisted of evaluating the gross energy requirements of fuels manufactured by the UK fuel supply industries during 1968 (Chapman, 1973b). These results, which are summarised in table 3.1, were used to assess direct energy inputs to all industrial sectors. The Census of Production statistics were then re-analysed using these intermediate results, indirect

Table 3.1 : Energy requirements of fuels produced in UK, 1968.

(Chapman, 1973b)

Fuel	Units	Energy content (MJ)	G.e.r.
Electricity*	kWhe	3.6	15
Coal	kg.	27	29
Coke	kg.	28	33
Derv fuel	litre	38	43
Fuel oil	kg.	43	49
Town gas	m^3.	18	24
Natural gas	m^3.	38	41

(* generated from fossil fuels)

energy inputs were determined and approximate energy intensities were deduced. Repeating this procedure with successively re-calculated inputs enabled these energy intensities to be refined and such iteration was only terminated when improvements became insignificant. Energy intensities of particular products and activities obtained by this analysis and used here are shown in table 3.2. To complete the statistical data base, important industrial sectors not covered by the Census of Production were also investigated and the results of the analysis of one particular sector, transport services, are illustrated in table 3.3 (Mortimer, 1974a, 1974b, 1975).

Although the numerical accuracy of results obtained from the Census of Production statistics has been estimated as ± 15% (Casper et al, 1974), they are probably less reliable since they mainly refer to diverse products and activities rather than individual, specifically-definable items. Because these results also contain aggregated information and are presented in financially-based units, they are normally used for 'order of magnitude' calculations and the evaluation of less important indirect energy inputs. Despite such limitations, however, the results of statistical analysis form a convenient basis for more detailed process analyses.

3.2 Results of process analysis

Numerous chemicals, metals and minerals are required in substantial amounts by nuclear fission power systems. Hence the evaluation of the total energy used by such systems involves the investigation of a wide range of ancillary industrial activities and the data base used here contains the energy requirements of many items examined by process analysis.

Table 3.2 : Selected energy intensities, UK 1968.

(Casper et al, 1974)

Census industrial sector	Energy intensity (MJ/£)	
	electrical	thermal
General iron and steel	35	620
Steel tubes	17	495
Pumps, valves and compressors	7	171
Industrial engines	7	216
Construction and earth-moving equipment	10	231
Mechanical handling equipment	8	186
Mining machinery	10	210
Industrial plant and steelwork	10	152
General mechanical engineering	20	178
Electrical machinery	9	182
Construction	3	115

Table 3.3 : Typical gross energy requirements of freight transport.

(Mortimer, 1974a, 1974b, 1975)

Mode	Gross energy requirement (MJ/te-km)	
	electrical	thermal
Light van		23
Heavy lorry		1.4
Diesel train		0.6
Electric train	0.12	0.2
Cargo ship		0.5
Super tanker		0.1
Aeroplane		14
Helicopter		44

In terms of the relative quantity consumed, sulphuric acid is the most important chemical input to burner reactor power systems and therefore all processes involved in its production were investigated in detail. Since the commonest method of manufacturing sulphuric acid is the catalytic oxidation of elemental sulphur, the initial step in this analysis consisted of examining the ways in which sulphur is extracted from natural raw materials.

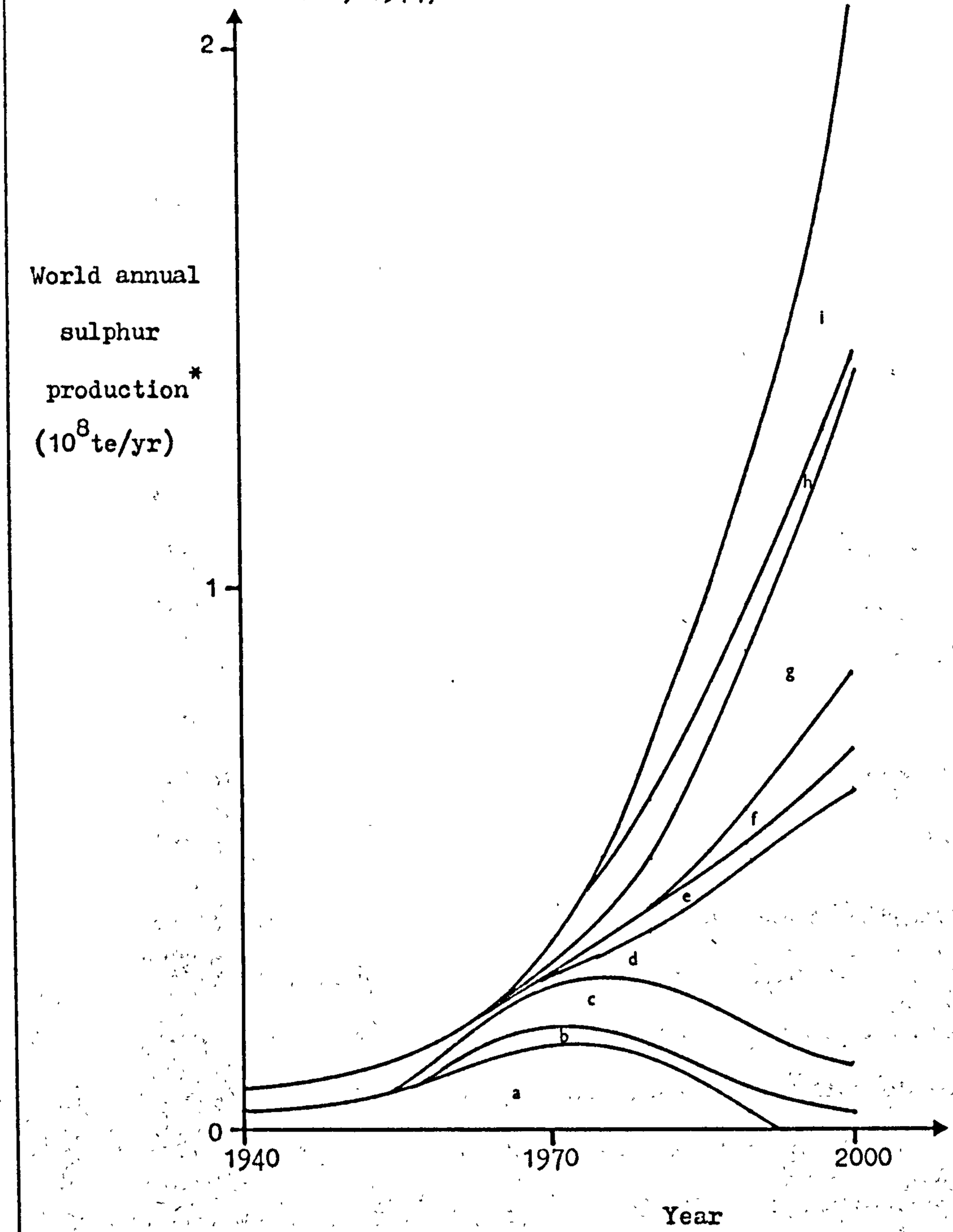
Sulphur can be obtained from a number of sources by a variety of methods and differences in production routes are reflected in resulting energy requirements which are summarised in table 3.4. Variations in the accessibility and concentration of sulphur in resources are partly responsible for the different values of energy requirement which extend from (-) 5 to (+) 150 MJ(t) per kg. The occurrence of negative energy requirements is due to the recovery of waste heat from certain operations which involve exothermic reactions. Error bars reflect, to some extent, the difference between best and worst practice in actual operations.

Since information on the particular origins of such a commonly-used chemical as sulphur is generally not available, it is not always possible to use these specific results in analysis. It is frequently more convenient to use a weighted average value based on the energy requirement and contribution to supply of each production route. Thus the average energy requirement of sulphur was deduced by combining the results of table 3.4 with data on the trends and forecasts of world sulphur production illustrated in figure 3.1. Fluctuations in the contribution of different methods to the total supply of sulphur cause a variation in the typical energy requirement

Table 3.4 : Energy requirements of sulphur.

Starting material	Production route	E.r. (MJ/kg) electrical	E.r. (MJ/kg) thermal
a) Salt-dome sulphur	Frasch process	-	11 ± 3.2
b) Sedimentary sulphur	Mining & flotation	0.8 ± 0.3	4.8 ± 2.6
c) Natural gas	Gas purification	0.5 ± 0.3	1.8 ± 7.4
d) Iron sulphide ores	Mining & smelting	(-) 6.1 ± 1.0	46 ± 7.4
e) Non-ferrous sulphides	Mining & smelting	0.8 ± 0.5	59 ± 38
f) Tar sand & oil shale	Desulphurisation	2.6 ± 0.3	120 ± 24
g) Coal	Desulphurisation	0.5 ± 0.1	53 ± 1.6
h) Oil	Desulphurisation	2.6 ± 5.3	85 ± 26
i) Sulphate minerals	Mining & roasting	-	29 ± 20
j) Sulphated brine	Purification	1.5	(-) 1.5 ± 1.6
k) Seawater	Ion extraction	0.8 ± 0.2	2.4 ± 1.8

Figure 3.1 : Trends in world sulphur production. (Fogarty and Mollison 1960; Engineering and Mining Journal, 1968; Harrer, 1969; Northolt, 1974)



(* excluding USSR and People's Republic of China)

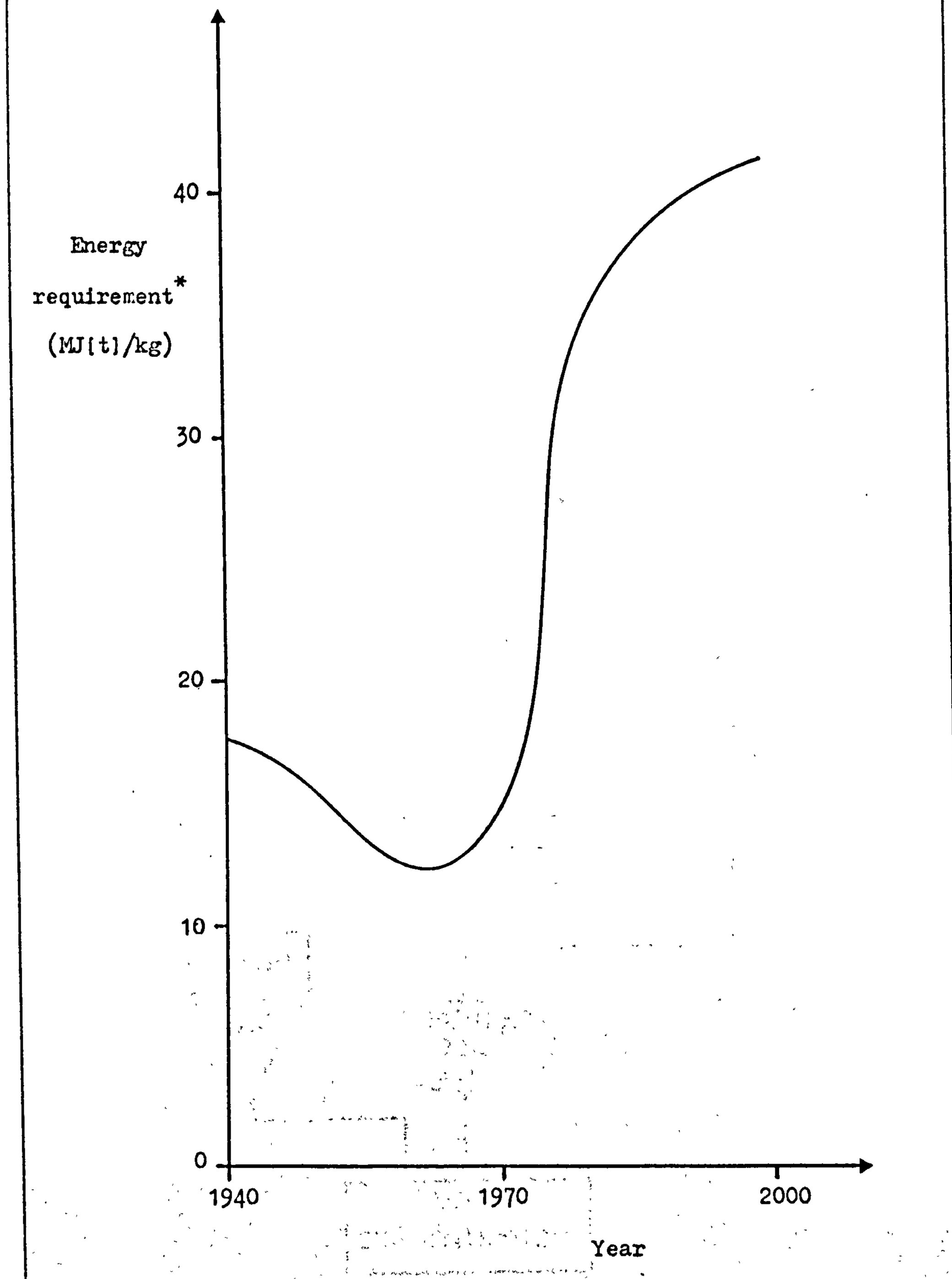
See table 3.4 for key to figure lettering.

which is shown in figure 3.2. This diagram indicates that the average sulphur energy requirement ranges from 12 to 42 MJ(t) per kg during the period in question.

Methods of producing sulphuric acid from sulphur, such as the important Frasch/Contact process route illustrated in figure 3.3, were investigated using these results. Other processes were also examined and all subsequent results, which extend from (-) 0.5 to (+) 41 MJ(t) per kg, are shown in table 3.5. Trends and forecasts of world sulphuric acid production, illustrated in figure 3.4, were used in conjunction with these data to deduce the weighted average energy requirement shown in figure 3.5. Although a value of (-) 1 MJ(t) per kg has been obtained by earlier work (Smith, 1969), this detailed analysis indicates that, during the particular period considered, the typical energy requirement varies between 4 and 10 MJ(t) per kg and these specific results were used here in the examination of the nuclear fuel cycle.

To complete the data base a number of processes which make other important materials required by the nuclear power system were also analysed using various sources of data and the results are given in table 3.6. The investigation of these processes was less extensive than the previous study of sulphuric acid production. However, since results incorporate error bars which reflect variations in natural conditions, industrial practice, etc., they were considered sufficiently reliable for this analysis of burner reactor power systems.

Figure 3.2 : World average energy requirement of sulphur.



(* assumes electricity generated by fossil-fired power stations;
4 MJ[t]/MJ[e])

Figure 3.3 : Flow diagram of the production of sulphuric acid from elemental sulphur using the Frasch/Contact process route.

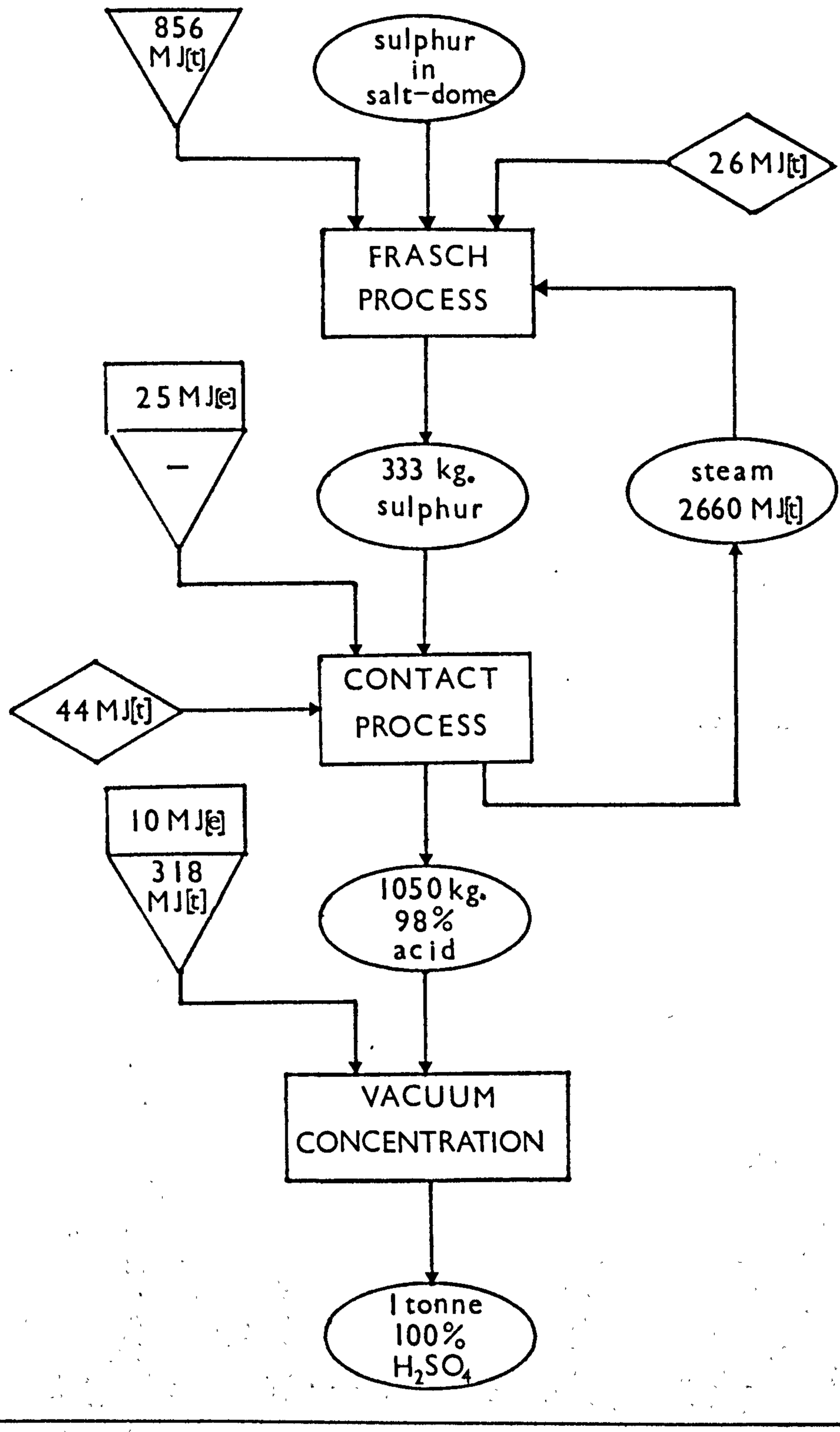


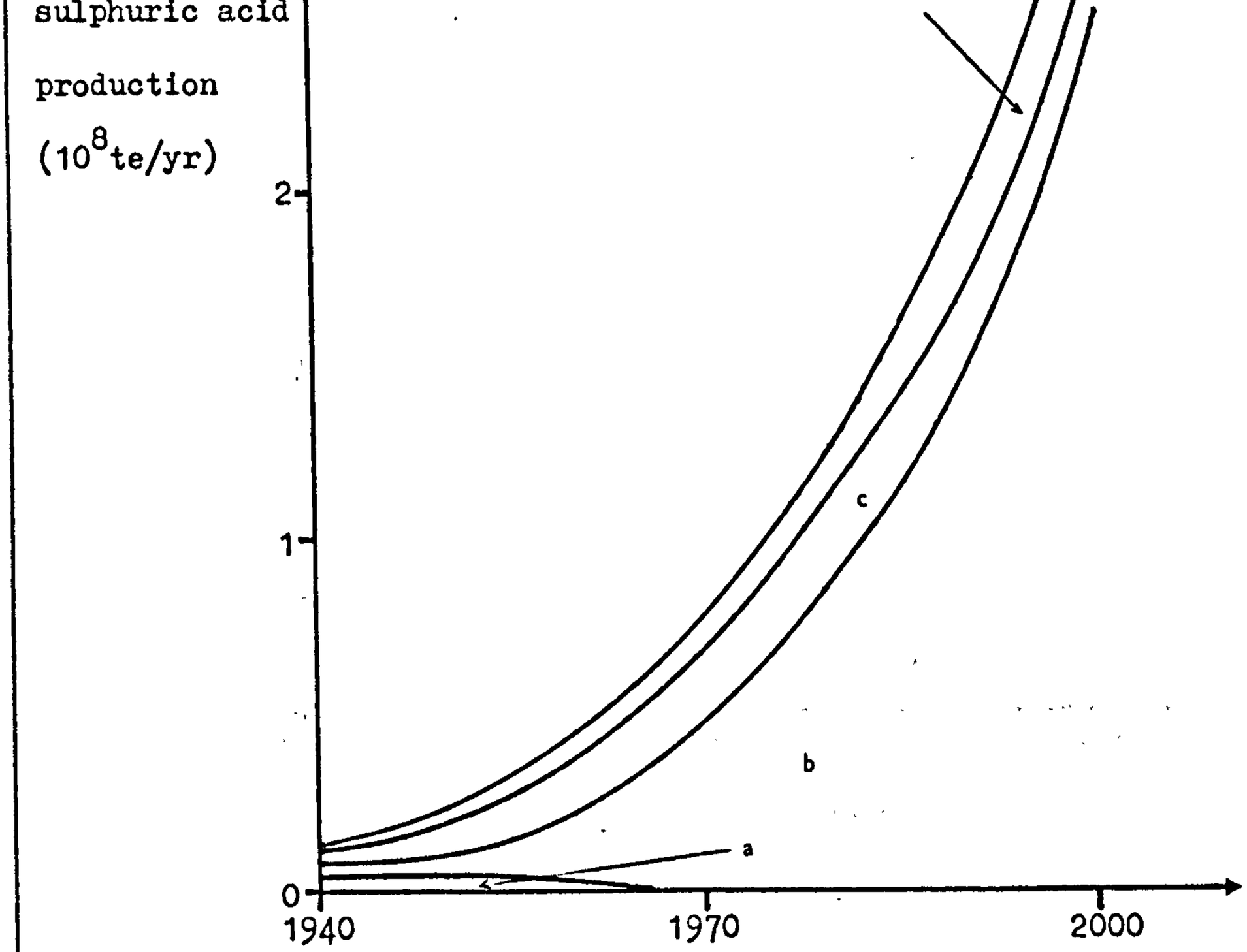
Table 3.5 : Energy requirements of sulphuric acid.

Starting material	Production process	E.r. (MJ/kg 100% acid)	
		electrical	thermal
a) Elemental sulphur	Lead chamber process	-	8.2±2.9*
b) Elemental sulphur	Contact process	-	2.6±2.9*
c) Iron sulphide ore	Contact process	0.4±0.1	(-)0.5±0.9
d) Smelter stack gas	Contact process	0.8±0.7	22 ±13
e) Boiler stack gas	Contact process	0.5	5.1±0.2
f) Anhydrite sulphate	Contact process	0.5±0.1	11 ±2.6
g) Gypsum sulphate	Contact process	1.0	9.7±7.4
h) Hydrogen sulphide	Contact process	0.2	2.0±0.2

(* assumes 1970 world average energy requirement for sulphur)

Figure 3.4 : Trends in world sulphuric acid production

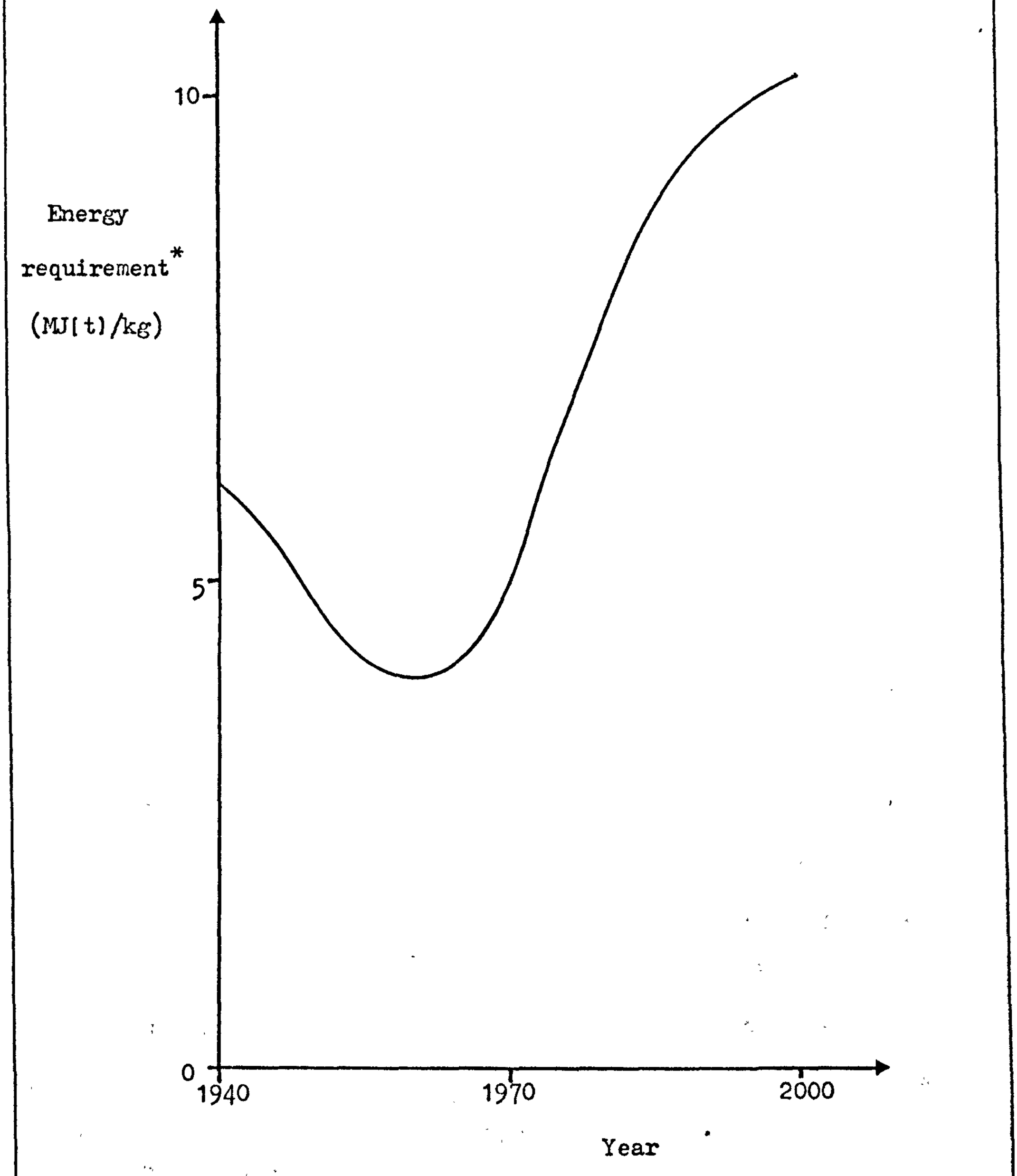
(Harrer, 1967 ; Zimmerman and Roman 1967; van Thoor 1968; Northolt 1974)



(* excluding USSR and People's Republic of China)

See table 3.5 for key to figure lettering.

Figure 3.5 : World average energy requirement of sulphuric acid.



(* assumes electricity generated by fossil-fired power stations;
4 MJ[t]/MJ[e])

Table 3.6 : Energy requirements of miscellaneous industrial materials

Material	Energy requirement (MJ/kg)	
	electrical	thermal
Aluminium sulphate[1,2]	0.15 ± 0.12	13 ± 2.0
Ammonia[1,2]	0.19 ± 0.04	22 ± 4.3
Ammonium nitrate[2]	0.37	25 ± 2.4
Calcium oxide[1,2]	0.07	8.4 ± 0.9
Cement (Portland)[1,3]	0.40 ± 0.04	6.3 ± 0.2
Deuterium oxide[4,5,6,7,8]	2200 ± 600	28000 ± 6200
Explosives (general)[3]	0.87 ± 0.40	70 ± 32
Fluorine[9]	64 ± 1	62 ± 17
Graphite (nuclear)[1,10,11]	30 ± 14	86 ± 27
Hydrofluoric acid[2]	4.3 ± 0.7	48 ± 12
Magnesium[1,11,12]	37 ± 10	170 ± 30
Nitric acid[1,2]	0.11 ± 0.02	12 ± 1.4
Sand and gravel[3]	0.02	0.14 ± 0.02
Sodium carbonate[1,2]	0.39 ± 0.06	27 ± 8.4
Sodium chloride[3]	0.01	0.31 ± 0.16
Sodium hydroxide[1,2]	5.8 ± 0.5	19 ± 0.1
Steel (mild)[13,14]	1.2	31 ± 11
Steel (stainless)[13,14]	18 ± 6	36 ± 12
Titanium hydroxide[12,15]	9 ± 7	3.8 ± 3.4
Water[3]	0.002	
Zirconium[11,16,17]	240 ± 55	300 ± 90

Initial data; (1) Shreve, 1967 (2) Faith, Keyes and Clark, 1965 (3) Casper et al, 1974 (4) Thayer and Bebbington, 1959 (5) Thayer, 1960 (6) Thayer and Proctor, 1962 (7) Benedict, 1955 (8) Barr and Drews, 1960 (9) Eumark and Sieghund, 1966 (10) Liggett and Bacon, 1964 (11) U.S. Bureau of Mines, 1975 (12) Bravard, Flora and Portal, 1972 (13) National Economic Development Office, 1974 (14) Waller, 1975 (15) Miller, J., 1957 (16) Miller, G., 1957 (17) Blumenthal and Roach, 1970.

4 BURNER REACTOR POWER SYSTEMS

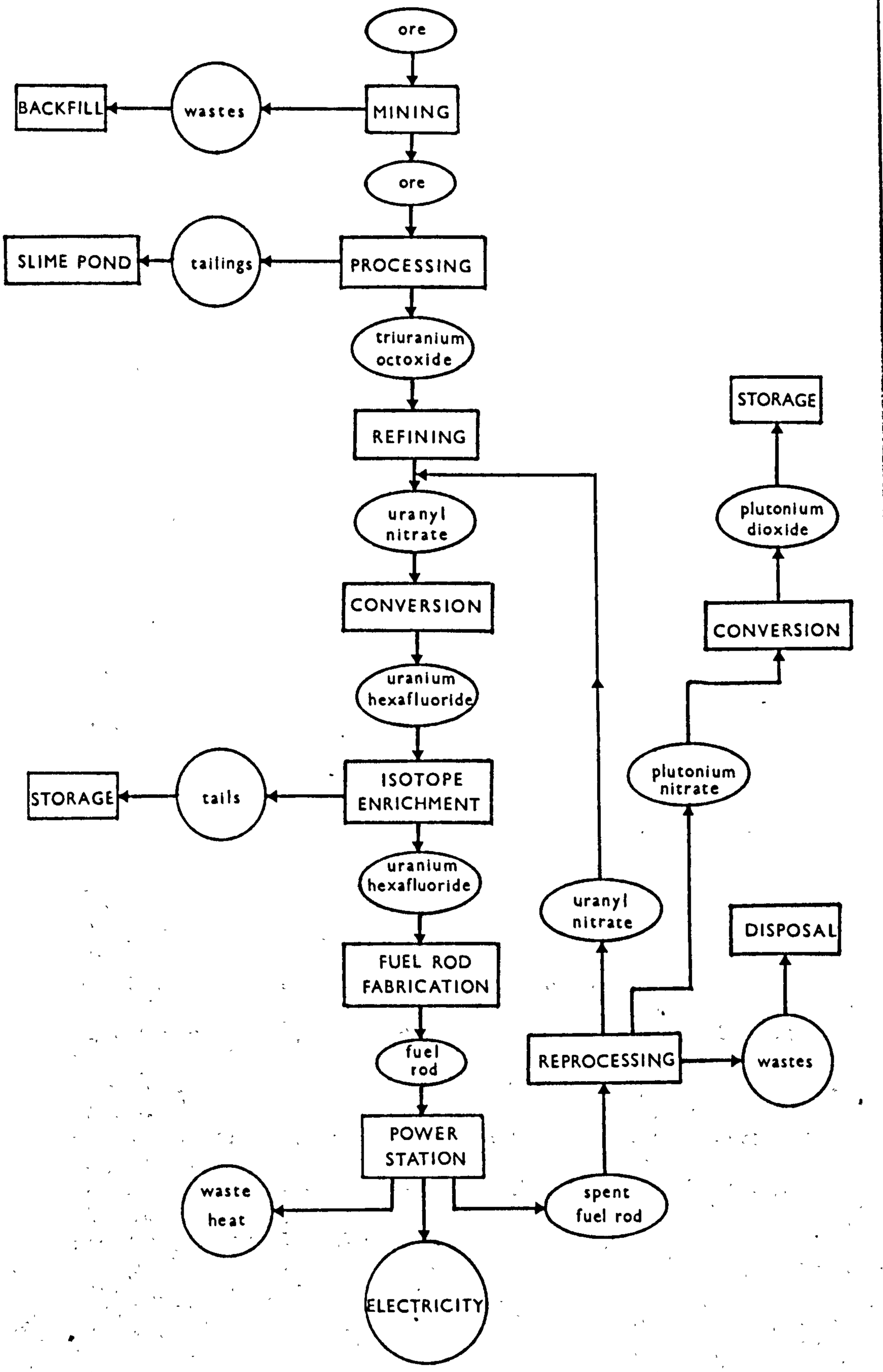
4.1 The fuel cycle

The burner reactor fuel cycle consists of a sequence of processes which are used to manufacture suitable fissile material, referred to as nuclear fuel, from naturally-occurring uranium and artificially-produced plutonium. The main features of a typical fuel cycle are illustrated in figure 4.1.

Uranium is the fundamental energy resource of burner reactor power systems and, although it can be found in many diverse materials such as coal and seawater, the most important commercial sources are ores which contain the native minerals pitchblende, uraninite, carnotite, tyuyamunite, uranophane, coffinite and autunite. The significance of these minerals is that they can occur in deposits which contain relatively high concentrations of uranium. The amount of useful material in an ore expressed as a fraction or percentage of the total material present is known as the ore grade and deposits of commercial uranium ore currently display grades which are a thousand times higher than those of common crustal rocks and a million times that of seawater.

Uranium ores are usually extracted from deposits using conventional underground and open-pit mining techniques. Underground mining is commonly applied to ore which occurs in relatively thin, continuous, deep-lying seams, whilst open-pit methods are generally used when the ore is contained in comparatively compact deposits close to the surface. Mining involves the extraction of ore by the complete or partial removal of material known as overburden which covers and

Figure 4.1 : Typical fuel cycle for a burner reactor using enriched fuel containing some recycled uranium.



surrounds the deposit. Physical properties of uranium minerals such as their characteristic radioactivity enable ore to be identified at the workface and distinguished from associated waste material which contains substantially less uranium. This ensures that ore produced by the mine contains as little worthless material as possible.

The next stage in the fuel cycle consists of processing the mined uranium ore in a mill plant to remove unwanted minerals known as gangue. This is achieved by comminuting, or crushing and grinding, the ore to reduce its particle size and then treating it with a suitable reagent such as sulphuric acid or sodium carbonate which extracts, or leaches, uranium from the ore. The resulting product consists of a uranium-bearing solution called leach liquor which can be separated from associated gangue by filtration and similar techniques that reject undissolved materials as wastes called tailings. A clarified uranium-rich liquid commonly referred to as the pregnant solution is obtained by concentrating the purified leach liquor and a complex mixture of uranium compounds known as uranium concentrate, or yellow cake, is produced from this by adding a reagent such as ammonia or sodium hydroxide which preferentially precipitates uranium salts. Specific details of the actual techniques and reagents used in such processing are mainly determined by the basic characteristics of the particular ore being treated. This also affects the composition of the final product obtained and the equivalent content of dried yellow cake can vary from 53% to 92% triuranium octoxide, U_3O_8.

Since yellow cake contains impurities which would interfere with fission reactions, it must be purified in a refinery prior to further progress in the fuel cycle. Refining consists of dissolving the yellow cake in nitric acid and selectively re-extracting pure uranyl nitrate, $UO_2(NO_3)_2 6H_2O$, with organic solvents. Uranium dioxide, UO_2, or black oxide as it is sometimes called, is obtained from this intermediate product by denitration in a conversion plant which is usually part of the refinery. Uranium dioxide is then powdered or reduced to uranium metal prior to fabrication into fuel rods for reactors which consume natural, or unenriched, uranium containing 0.71% U-235. Since certain other types of reactor use fuel containing enriched uranium with a higher proportion of U-235 than the natural abundance, some fuel cycles incorporate a process known as isotope enrichment prior to fuel fabrication. Conventional enrichment techniques use uranium hexafluoride gas, UF_6, commonly referred to as hex, and this is produced in the conversion plant by reacting uranium dioxide with hydrofluoric acid and fluorine.

Enrichment involves increasing the abundance of one particular isotope in a mixture of isotopes. In addition to the manufacture of an artificially-enriched output, enrichment processes also produce material, often known as the depleted fraction, which contains a smaller amount of the given isotope than the original natural mixture. For uranium enrichment in particular material depleted in U-235 occurs in wastes called tails.

The gas diffusion technique is currently the most widespread method used to enrich uranium. This consists of pumping

uranium hexafluoride gas through a series of porous membranes. The basic principle is that the mass difference between the isotopes U-235 and U-238 influences the rate at which isotopically different gases diffuse through the membranes. This effect enables the concentration of the lighter isotope in the gas stream to be progressively enhanced. Alternative enrichment processes are currently being examined and developed, and these include, in particular, the gas centrifuge method which uses differences in centrifugal forces acting on isotopically dissimilar molecules in a rapidly rotating container to separate isotopes. Gas enriched in U-235 is collected near the axis of the centrifuge. Other experimental techniques being studied include jet nozzle and vortex tube methods, plasma centrifuge processes and laser devices.

The next stage in the fuel cycle consists of fabricating suitable nuclear material into fuel rods which can be placed in reactors to provide energy. Natural uranium dioxide powder and uranium metal ingots obtained from the refinery can be directly formed into slugs or pellets at the fabrication plant, whilst enriched uranium hexafluoride from the enrichment plant must first be re-converted to uranium dioxide. All types of fuel slugs and pellets are then made into fuel rods by surrounding them with a protective sheath of cladding material. This guards the nuclear fuel from corrosion and oxidation, and also provides containment for isotopes known as fission products formed by reactions that occur whilst the rod is in the reactor. Fabrication is completed by assembling fuel rods into an array called a fuel element which can be individually inserted into, or withdrawn from, the reactor.

Since the accumulation of fission products can cause distortion and uncharacteristic thermal effects, fuel rods are usually removed from the reactor well before all the U-235 has been consumed by fission. Important fissile materials such as unreacted uranium and plutonium produced by conversion reactions are recovered from this so-called spent fuel by reprocessing. Reprocessing consists of removing the cladding from the fuel rod either mechanically or chemically and dissolving the spent fuel in nitric acid. The resulting solution is clarified and treated with reagents that selectively extract uranium and plutonium. Uranyl nitrate is recovered in this way and, since it is usually depleted in U-235, converted to uranium hexafluoride for enrichment prior to its re-introduction into fuel rods. Plutonium nitrate is generally converted to plutonium dioxide and either stored for subsequent use in breeder reactors or incorporated into burner reactor fuel rods. Specific details of the re-introduction, or re-cycling, of fissile materials depend on the particular fuel management policy adopted.

Wastes produced from reprocessing contain unwanted fission products such as americium, caesium, curium, neptunium, strontium, etc., and unrecovered uranium and plutonium. These wastes can be categorised into three general groups depending on the amount and nature of the radioactive material they contain. These categories are referred to as low-, intermediate- and high-level wastes. Low- and intermediate-level wastes contain small amounts of radioactive and contaminated material and are usually buried or discharged into the sea. High-level wastes contain highly radiotoxic isotopes

which must be carefully isolated from the environment. At the moment such wastes are stored in large tanks whilst work continues in the search for a permanent solution to their disposal.

This concludes the brief description of the processes involved in the production of nuclear material for burner reactor power systems. Figure 4.2 illustrates some further details of a typical fuel cycle. More comprehensive accounts can be found in other literature such as the summaries presented by A. B. Cambel (1965), V. L. Mattson (1970), E. A. Youngberg (1973) and Sir John Hill (1974).

4.2 The nuclear power station

A nuclear power station is an arrangement of equipment and machinery used to initiate, sustain and control nuclear fission reactions that release energy which is subsequently converted into a useful form of fuel, typically electricity, or sometimes process steam. The basic features of a typical burner reactor power station are illustrated in figure 4.3.

The reactor is a piece of equipment which provides the means of sustaining and controlling fission reactions which occur in the fuel elements. These fuel elements contain nuclear material such as uranium metal, and uranium dioxide and plutonium dioxide powder, and are arranged in a pre-determined pattern at the centre, or core, of the reactor. In addition to fuel elements the core also contains moderator material which slows down neutrons produced by fission reactions and therefore enhances the likelihood of further fission reactions. The rate of the nuclear reaction in the core is regulated by withdrawing

Figure 4.2 : SGHWR fuel cycle incorporating uranium recycling.

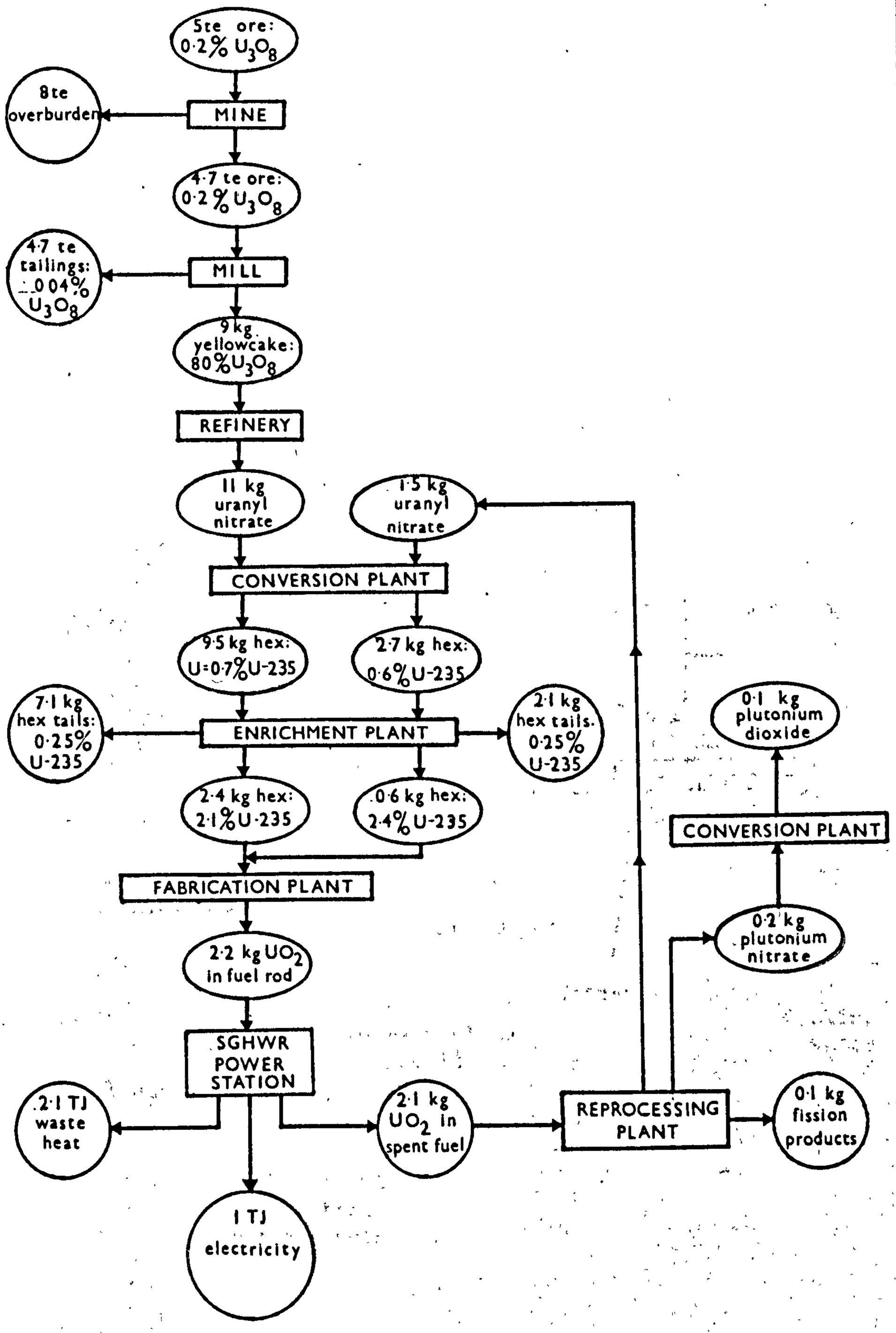
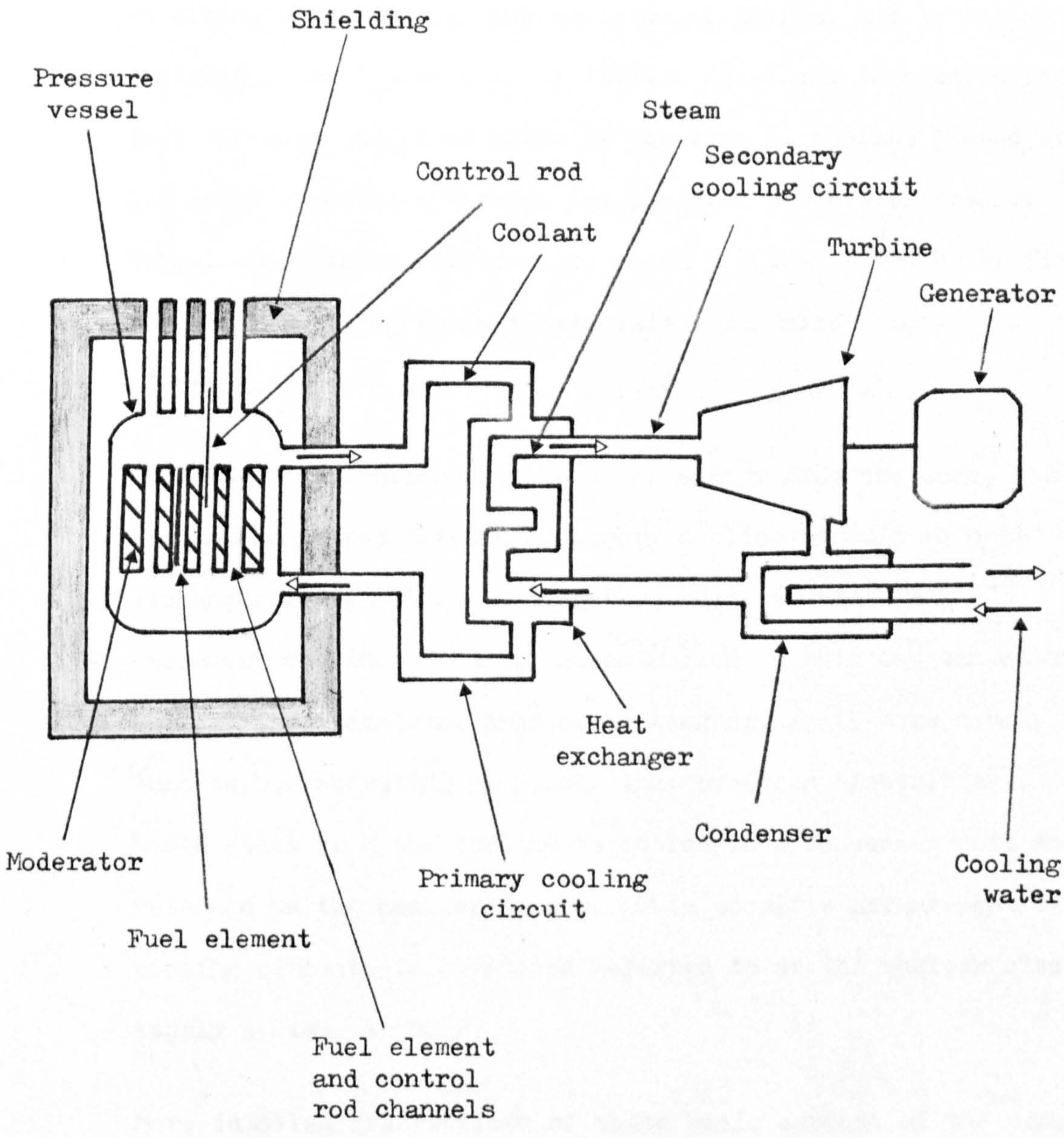


Figure 4.3 : General schematic layout of a burner reactor power station.



or inserting control rods which are made of materials that strongly absorb fission-producing neutrons.

In some reactor designs the core is situated in a pressure vessel, or containment chamber, whilst in others the fuel elements are contained in pressure tubes. In either case the contained reactor core is surrounded by a composite wall of shielding which reduces the emission of thermal and ionising radiation. Heat generated by fission reactions is removed from fuel elements and other parts of the core by coolant liquid or gas which circulates through the pressure vessels or pressure tubes. Certain reactor designs avoid the need for a bulky fixed moderator by using coolant materials which also moderate neutrons.

To ensure the continuous removal of energy from the core, hot coolant is pumped through a primary cooling circuit to heat exchanger equipment which transfers heat to water in a secondary cooling circuit. Steam raised in this way can either be used for industrial process heating or, as is more usual, to turn turbo-generating machinery that produces electricity. Waste steam from the turbine is cooled in a condenser unit and returned to the heat exchanger. This complete arrangement of cooling circuits is sometimes referred to as the nuclear steam supply system, or NSSS.

More detailed descriptions of these basic aspects of the burner reactor power station can be obtained from the literature of other authors, for example, S. E. Hunt (1974) and W. C. Patterson (1976). Although general features are shared by all burner reactors, there are important differences between the designs which are currently available. Specific features of

the particular reactor types investigated here are listed in table 4.1.

The six commercial reactor designs examined have the abbreviated titles; MAGNOX, AGR, SGHWR, CANDU, BWR and PWR. The world's first civil nuclear power station was based on the British-designed Magnox reactor which takes its title from the special magnesium alloy called magnox used to clad its fuel. The UK's second programme design was the Advanced Gas-cooled Reactor, or AGR, and a candidate for the third series is the Steam Generating Heavy Water Reactor, or SGHWR, which uses deuterium oxide, or heavy water, as a moderator. The CANadian Deuterium Uranium, or CANDU, design uses heavy water not only as a moderator but also as a coolant. Designs developed in the USA consist of Light Water Reactors, or LWR's, which use ordinary, or light, water as a combined coolant and moderator. These general designs include the Boiling Water Reactor, or BWR, and the Pressurised Water Reactor, or PWR. The main difference between these reactors is that cooling water is allowed to boil in the core of the BWR, whilst high pressures are maintained in the PWR to prevent this.

Thorough descriptions of all these types of burner reactor are available in the accounts of other authors who include, in particular, E. Gabriel and D. Smith (1970), E. S. Booth (1971), Sir John Hill (1971), S. E. Hunt (1974) and W. C. Patterson (1976).

Table 4.1 : Features of burner reactors

Design	Fuel	Cladding	Moderator	Coolant	Containment
MAGNOX	Natural uranium metal	Magnox	Graphite	Carbon dioxide gas	Steel pressure vessel
AGR	Enriched uranium dioxide	Stainless steel	Graphite	Carbon dioxide gas	Concrete pressure vessel
SGHWR	Enriched uranium dioxide	Zirconium	Deuterium oxide	Water	Zirconium pressure tubes
CANDU	Natural uranium dioxide	Zirconium	Deuterium oxide	Deuterium oxide	Zirconium pressure tubes
BWR	Enriched uranium dioxide	Zirconium	Water	Water	Steel pressure vessel
PWR	Enriched uranium dioxide	Zirconium	Water	Water	Steel pressure vessel

5 ANALYSIS OF THE NUCLEAR FUEL CYCLE

5.1 Introduction

The purpose of this chapter is to estimate the total amount of energy required to manufacture the type of nuclear fuel used in burner reactors. This involves investigating each stage of the nuclear fuel cycle. To clarify calculation of the energy requirement of nuclear fuel, analysis consists of formulating an expression which contains energy input terms that describe each part of the fuel cycle. In many causes these terms are composed of a number of basic parameters which represent fundamental aspects of operations and processes. This approach enables the energy requirement of nuclear fuel to be evaluated under different conditions and allows the influence of important factors to be assessed. Further details of this analysis are presented in the relevant appendices.

5.2 Production of uranium concentrate from ore

5.2.1 Exploration

The first step in the production of uranium consists of locating commercial sources of ore. Ore can occur in deposits of differing geological structure, extent, composition and grade. This is reflected in the variety of exploration activity required to discover economically-viable deposits. Hence exploration can consist of simple prospecting with radioactive detection devices or extensive surveying which may involve aerial mapping, sampling and core drilling.

Such diversity of operations obviously affects the energy input to this initial stage of the fuel cycle. Scarcity of

specifically detailed information also limits analysis to the evaluation of 'order of magnitude' estimates of the energy requirement. Consequently typical modern exploration programmes, consisting of airborne and surface reconnaissance, were investigated and the following equation was established to describe the total energy required by such activity;

E_1 = energy input per unit mass U_3O_8 discovered

$$= (e_{1a} \times d_{1a}) + (e_{1b} \times d_{1b}) + (e_{1c} \times d_{1c})$$

where,

e_{1a} = g.e.r. of exploratory drilling

e_{1b} = g.e.r. of air transport

e_{1c} = g.e.r. of private and freight road transport

d_{1a} = distance drilled per unit mass U_3O_8 discovered

= exploration drilling rate

d_{1b} = total distance flown per unit mass U_3O_8 discovered

d_{1c} = total distance travelled by car and truck per unit mass U_3O_8 discovered

Only these particular items are included in the energy equation because supporting work such as laboratory assaying, etc., uses significantly less energy than drilling and transport operations.

The energy requirement of exploratory drilling was obtained by studying the use of fuel and other items by various drilling equipment operating under a wide range of conditions. Details of this analysis are given in appendix A. Numerous fundamental factors influence both the direct and indirect energy requirements of drilling and the effect on the g.e.r., e_{1a}, is reflected in the minimum and maximum values illustrated in

table 5.1. The energy requirements of transportation, e_{1b} and e_{1c}, obtained from the data base (Mortimer, 1974a, 1975), are also shown in this table.

Other parameters incorporated in the equation relate exploration to discovery. The exploration drilling rate, d_{1a}, was deduced from mining statistics (Woodmansee, 1970; Colorado School of Mines, 1973; Johnson, 1974) and recent values for the USA are illustrated in figure 5.1. The transport factors, d_{1b} and d_{1c}, given in table 5.1 were found by examining a number of exploration case studies (Bowie, Davis and Ostle, 1972).

The values given in table 5.1 were used to estimate the current amount of energy required to find a commercial deposit of conventional uranium ore and results revealed that the range can extend from 10^2 to 10^5 MJ(t) per tonne U_3O_8.

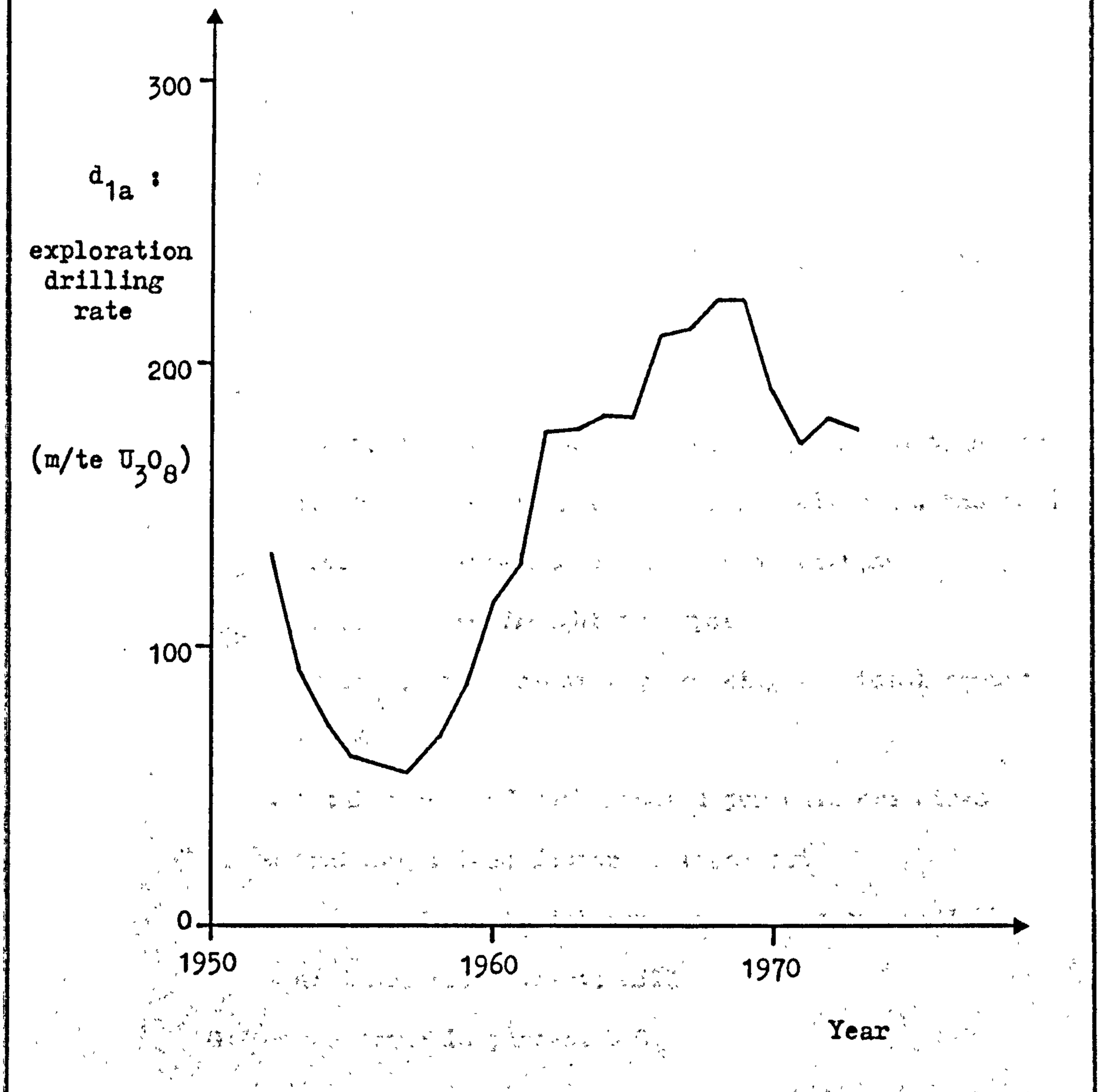
5.2.2 Conventional ore mining

For the purpose of energy analysis conventional ore mining can be divided into four general types of operation. The first is rock breaking which involves freeing material from the ground and reducing its size to suitable proportions for further handling. Rock breaking is usually achieved by drilling and blasting, although in soft ground such activity can often be avoided. The next operation is rock excavation in which broken ore and other material is removed from the site of mining, or workface, by machinery such as power shovels, bulldozers, scrapers, conveyors, etc. Waste material is usually hauled to dumping areas, whilst ore is transferred to storage piles. The third category of operations comprises all other activities in the mine which consume appreciable amounts of fuel such as

Table 5.1 : Basic parameters for exploration

Parameter	Units	Minimum	Maximum
e_{1a}	MJ(t)/m	10	1000
e_{1b}	MJ(t)/km	65	110
e_{1c}	MJ(t)/km	5	25
d_{1a}	m/te U_3O_8	50	200
d_{1b}	km/te U_3O_8	0	100
d_{1c}	km/te U_3O_8	0	10

Figure 5.1 : US average exploration drilling rate for deposits containing ore with grades greater than 0.1% U_3O_8.



ventilation, heating/cooling, water drainage, etc. The remaining operation consists of transporting the ore from the mine to the next stage in the fuel cycle, the ore processing mill.

All these operations influence the total amount of energy required to extract ore from the ground and the following equation was formulated to demonstrate this;

E_2 = energy input per unit mass U_3O_8 mined and delivered to the mill

$$= \left[\left(\left\{ e_{2a} \times t \right\} + e_{2b} + e_{2c} \right) \times w + \left(e_{2d} \times \frac{d_2}{l_2} \right) \right] \times \frac{100}{G}$$

where,

e_{2a} = g.e.r. of rock breaking (i.e. drilling and blasting)

e_{2b} = g.e.r. of rock excavation (i.e. loading and hauling)

e_{2c} = g.e.r. of miscellaneous mining operations

e_{2d} = g.e.r. of ore freight transport

t = amount of rock requiring breaking per total amount mined

w = total amount of rock handled per unit ore mined

l_2 = fractional load factor of transport

= amount of ore loaded per total capacity available

d_2 = distance from mine to mill

G = ore grade in percent U_3O_8

The g.e.r. of rock breaking, e_{2a}, equals the total energy input to drilling and blasting per unit material, ore, overburden or waste, treated. This depends on various basic factors which are generally determined by the type of mining used - underground or open-pit. Variations in the g.e.r., e_{2a}, which

was based on results presented in appendix A, for both forms of mining are illustrated in tables 5.2 and 5.3. The g.e.r. of rock excavation, e_{2b}, equals the total energy input to loading and hauling per unit material handled. The g.e.r. of miscellaneous mining operations, e_{2c}, represents the effective energy input from ancillary activity for each unit of material mined. The g.e.r. of freight transport, e_{2d}, indicates the total energy required to carry one unit mass of ore for one unit of distance. Results of the study of direct and indirect energy inputs to rock excavation, miscellaneous mining operations and ore transportation are given in appendix B and ranges of the resulting g.e.r. values for different types of mining are shown in tables 5.2 and 5.3.

The fraction of rock requiring drilling and blasting, t, depends on ground conditions and, as the tables indicate, this can vary from 0 to 1. The ratio of material handled to ore mined, w, represents the sum of two important mining parameters, the stripping ratio and the waste ratio;

$$w = 1 + \text{stripping ratio} + \text{waste ratio}$$

The stripping ratio equals the amount of overburden removed per unit ore extracted, whereas the waste ratio is the amount of low-grade, barren, or waste, material handled at the workface per unit ore mined. In very rich, shallow and compact deposits these parameters can fall to zero and w equals one. As tables 5.2 and 5.3 indicate, current values of w can be as high as 5 for underground mines and 60 for open-pit operations.

The load factor for transport, l_2, relates the amount of freight carried to the actual capacity available. Since most outward

Table 5.2 : Basic parameters for underground mining

Parameter	Units	Minimum	Maximum
e_{2a}	MJ/te material	1.3(e) + 27.0(t)	46.0(e) + 200(t)
e_{2b}	MJ/te material	6.7(e) + 35.0(t)	19.0(e) + 170(t)
e_{2c}	MJ/te material	4.3(e) + 3.7(t)	17.0(e) + 170(t)
e_{2d}	MJ/te-km	0.2(t)	3.1(e)
t	-	0	1
w	te/te ore	1	5
$\frac{1}{l_2}$	-	2	2
d_2	km	0.8	240

Table 5.3 : Basic parameters for open-pit mining

Parameter	Units	Minimum	Maximum
e_{2a}	MJ/te material	0.2(e) + 4.3(t)	2.7(e) + 24.0(t)
e_{2b}	MJ/te material	1.1(e) + 1.3(t)	9.3(e) + 3.5(t)
e_{2c}	MJ/te material	0.4(e)	10.0(e)
e_{2d}	MJ/te-km	0.2(t)	3.2(e)
t	-	0	1
w	te/te ore	1	60
$\frac{1}{l_2}$	-	2	2
d_2	km	0.8	240

journies are completed with full loads and return trips are usually made with no cargo, the load factor generally averages 0.5. The distance travelled from the mine to mill, d_2, can vary over a very wide range and the values shown in tables 5.2 and 5.3 were deduced from geographical data.

The full extent of the possible variation of the total energy input to conventional underground and open-pit uranium mining which results from the combination of these various factors is illustrated in table 5.4. Assuming a conversion factor of 4 MJ(t) per MJ(e) for electricity, it can be seen that the lowest energy input is approximately 10 MJ(t) per tonne ore mined and the highest is 10000 MJ(t) per tonne ore. Analysis of individual mines indicates a smaller range of 400 MJ(t) to 2500 MJ(t) per tonne ore. The difference between these ranges arises because theoretically calculated results are intended to cover all possible combinations of mining conditions whereas the observed values reflect the fact that, in practice, the extremes of all conditions are rarely encountered simultaneously.

5.2.3 Conventional ore processing

There are many ways of producing uranium from natural sources. Although techniques used depend on the characteristics of the particular source in question, most common ores can be treated by relatively similar methods, collectively referred to here as conventional ore processing. This generally consists of comminuting, or pulverising, the ore, leaching, or washing, the resulting material with an appropriate reagent and recovering uranium as a complex concentrate containing triuranium octoxide, or U_3O_8, from the subsequent solution. Such methods are used

Table 5.4 : Energy input to conventional uranium mining

Type of mining		Energy input: E_2 (MJ/te U_3O_8) electrical		thermal
Underground	- minimum :	$\frac{1100}{G}$	+	$\frac{3900}{G}$
	maximum :	$\frac{190000}{G}$	+	$\frac{270000}{G}$
Open-pit	- minimum :	$\frac{150}{G}$	+	$\frac{160}{G}$
	maximum :	$\frac{285000}{G}$	+	$\frac{165000}{G}$

to treat currently-commercial ores and they could also be used to extract uranium from less orthodox sources.

The energy inputs to conventional ore processing can be divided into three categories; the energy required to produce concentrate from ore, dry it, and transport it from the mill to the refinery which is the next step in the fuel cycle. These inputs are summarised by the following expression which was formulated during analysis;

E_3 = energy input per unit mass U_3O_8 processed and delivered to the refinery

$$= (e_{3a} \times \frac{100}{r_3 \times G}) + (e_{3b}) + (e_{3c} \times \frac{d_3}{l_3})$$

where,

e_{3a} = g.e.r. of producing wet concentrate from ore

e_{3b} = g.e.r. of drying wet concentrate

e_{3c} = g.e.r. of concentrate freight transport

G = ore grade in percent U_3O_8

r_3 = fractional uranium recovery efficiency of ore processing

= amount of U_3O_8 produced in concentrate per amount of U_3O_8 processed in ore

l_3 = fractional load factor of transport

= amount of U_3O_8 carried per total capacity available

d_3 = distance from mill to refinery

The energy required to produce unfinished concentrate from ore, e_{3a}, the fuel consumed in drying, e_{3b}, and the fraction of uranium recovered from the initial feed during processing, r_3, all depend on the type of ore treated and the processing

technique used. The energy input to conventional ore processing, E_3, is influenced by these parameters and the full variation of this item was deduced by identifying those processes which consume the least and most energy. Evaluation of these parameters is described in appendix C and the combinations which produce minimum and maximum values of E_3 are shown in table 5.5. The minimum values correspond to the processing of uranium-bearing lignites under the most favourable operating conditions, whilst the maximum values represent the worst case for the treatment of uraniferous leached-zone phosphate clays (see appendix C). Values of e_{3a}, e_{3b} and r_3 for the processing of other ores combine to give results which lie between these particular extremes.

In addition to showing parameters which refer to ore processing in particular, table 5.5 also illustrates the ranges of the energy requirement of freight transport, e_{3c}, the load factor, l_3, and the distance between mill and refinery, d_3. The energy required for transportation was obtained from the data base and the range shown corresponds to results for various forms of rail freight transport. Load factors deduced from operating data equal the product of the fractional capacity factor (cf. appendix B) and the proportion of U_3O_8 in the concentrate. The fractional capacity factor relates the actual load carried on the outward and return journey to the amount of space available and generally varies from 0.5 to 1.0. The proportional amount of U_3O_8 in the concentrate indicates the purity of the mill product and, for the processes examined, ranges from 80% to 100% U_3O_8. Estimates of the distance that concentrate is transported were obtained from geographical information.

Table 5.5 : Basic parameters for conventional uranium ore processing

Parameter	Unit	Minimum	Maximum
e_{3a}	MJ/te ore	45(e) + 150(t)	165(e) + 2300(t)
e_{3b}	MJ/te U_3O_8	5(t)	552000(t)
e_{3c}	MJ/te-km	0.6(t)	0.3(e) + 0.3(t)
$\frac{1}{r_3}$	-	1.14	2.38
$\frac{1}{l_3}$	-	1.25	2.00
d_3	km	2	650

Using the parameters given in table 5.5 the full variation of the energy input to conventional ore processing, E_3, was deduced. Table 5.6 illustrates the minimum and maximum values for this stage of the fuel cycle. The energy required to produce uranium concentrate by the conventional treatment of any ore should fall within these limits. At the moment, with an electrical conversion factor of approximately 4 MJ(t) per MJ(e) and ore grades of around 0.2% U_3O_8, these limits stretch from 2×10^5 to 4×10^6 MJ(t) per tonne U_3O_8. Analysis of individual mills treating 0.2% U_3O_8 grade ore indicates that the current average energy input for processing is 8×10^5 MJ(t) per tonne U_3O_8.

5.2.4 Summary of uranium concentrate production

Uranium concentrate, or yellow cake, containing a high proportion of triuranium octoxide, U_3O_8, is the basic product of the initial stages of the nuclear fuel cycle. All nuclear fuel used in burner reactor power stations is primarily obtained from uranium concentrate which can be produced by numerous techniques from various sources. The energy required to recover uranium, by currently conventional means from common orthodox ores, equals the sum of energy inputs to exploration, mining and ore processing. This can be represented by the following equation;

E_u = net energy requirement of uranium concentrate

$$= \frac{E_1}{r_2 \times r_3} + \frac{E_2}{r_3} + E_3 \qquad \text{MJ/tonne } U_3O_8$$

Table 5.6 : Energy input to conventional uranium ore processing

Value	Energy input : E_3 (MJ/te U_3O_8)		
	electrical		thermal
Minimum :	$\left(\frac{5130}{G}\right)$	+	$\left(\frac{17100}{G} + 6.5\right)$
Maximum :	$\left(\frac{39300}{G} + 390\right)$	+	$\left(\frac{547000}{G} + 553000\right)$

where,

E_1 = energy input to exploration per unit mass U_3O_8 found

E_2 = energy input to mining per unit mass U_3O_8 mined and delivered to the mill

E_3 = energy input to ore processing per unit mass U_3O_8 processed and delivered to the refinery

r_2 = fractional uranium recovery efficiency of ore mining
= amount of U_3O_8 recovered by mining per amount of U_3O_8 in the deposit

r_3 = fractional uranium recovery efficiency of ore processing
= amount of U_3O_8 produced in concentrate per amount of U_3O_8 in ore

Minimum and maximum values of the energy inputs E_1, E_2 and E_3 are given in previous sections and estimates of the recovery factors r_2 and r_3 are shown in table 5.7. The resulting range of the net energy requirement for uranium concentrate, E_u, is illustrated in table 5.8 and the variation of E_u with ore grade, G, assuming an electricity conversion factor of 4 MJ(t) per MJ(e), is presented in figure 5.2. This partly derived variation can be compared with empirical data points for actual and proposed mines and mills. These data points were obtained by analysing operating information for individual plants and results show that the equation adequately contains actual fluctuations between its lower and upper bounds.

The scatter of data points illustrated in figure 5.2 is largely due to the variation of ore grades. The energy inputs to proposed projects (eg. Hurst, Crouse, Brown and Ross, 1966; Bieniewski, Persse and Brauch, 1971) are higher than those of

Table 5.7 : Recovery factors for uranium mining and ore processing

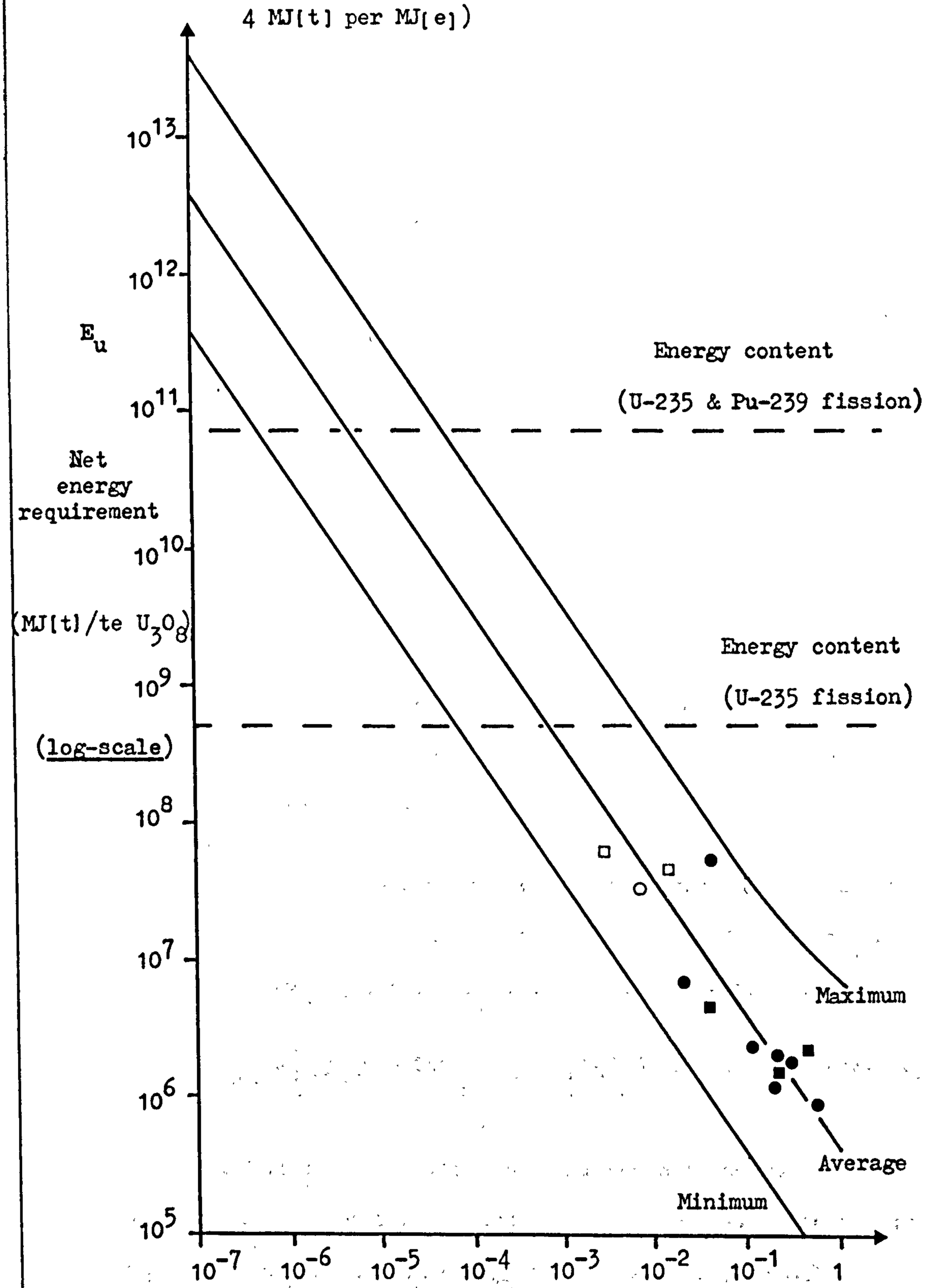
Parameter	Minimum	Maximum
$\frac{1}{r_2}$	1.25	5.00
$\frac{1}{r_3}$	1.14	2.38

Table 5.8 : Energy requirement of uranium concentrate

Value	Energy requirement : E_u (MJ/te U_3O_8)		
	electrical		thermal
Minimum :	$\frac{(5300)}{G}$	+	$\left(\frac{17300}{G} + 720\right)$
Maximum :	$\left(\frac{718000}{G} + 390\right)$	+	$\left(\frac{940000}{G} + 3070000\right)$

Figure 5.2 : Net energy requirement of uranium concentrate

(assuming an electrical conversion factor of 4 MJ[t] per MJ[e])



Operational plants;
with underground mine = ●
with open-pit mine = ■

Proposed plants;
with underground mine = ○
with open-pit mine = □

current operations because they are intended for the treatment of low grade ores which may be exploited in the future. Differences between the energy required by currently-commercial mines and mills which process similar grade material can occur, however, and fluctuations can amount to ± 50%. This is caused by differences in mining and processing conditions and techniques. Operations which consume subtantially more fuel than others can still be economically competitive, however, since fuel costs are only a small part of total costs for high grade ores (generally less than 20% for grades higher than 0.1% U_3O_8). Factors other than ore grade and energy requirement can influence costs and since total costs determine the economic viability of an operation, contemporaneously workable ore grades can extend over a very wide range. For example, although the average grade of ores that could be treated for $ 8 per pound U_3O_8 in 1972 in the USA was 0.2% U_3O_8, the full range varied from 0.04% to over 0.5% U_3O_8 (U. S. Atomic Energy Commission, 1972).

The energy equation and data points for uranium concentrate production illustrated in figure 5.2 can be used to obtain an average energy requirement represented by the expression;

$$(E_u)_{average} = \left[\frac{65000}{G} + 200\right](e) + \left[\frac{140000}{G} + 47000\right](t) \text{ MJ/te } U_3O_8$$

The use of an average variation can frequently be more convenient in subsequent analysis. Additional evidence for this type of logarithmic mean energy requirement is partially provided by statistical information concerning the actual distribution of basic mining parameters such as the stripping ratio, waste ratio, deposit depth, seam thickness, etc., which

affect energy inputs (Woodmansee, 1970; U. S. Atomic Energy Commission, 1972; U. S. Bureau of Mines, 1972).

For the sake of illustration, the minimum, maximum and mean energy inputs to uranium concentrate production can all be compared to the theoretical energy output, or energy content, of uranium. The total heat available from the complete fission of all U-235 nuclei in one tonne of U_3O_8 is 5×10^8 MJ(t) and this is the ultimate amount of energy that can be produced by this particular material in a burner reactor. In addition to U-235 fission, a breeder reactor can utilise the conversion of U-238 and subsequently promote the fission of Pu-239. Assuming a perfectly efficient breeding cycle, the maximum amount of energy that could be released is 7×10^{10} MJ(t) per tonne U_3O_8. In terms of available heat, these values determine the maximum amount of energy that can be consumed in the use of uranium as a fuel.

Although energy is consumed in subsequent parts of the nuclear fuel cycle and in nuclear power station construction and operation, the production of uranium concentrate from natural sources is regarded, in this brief investigation, as the most fundamental component of the burner reactor power system. Hence, assuming that the energy required to manufacture uranium concentrate should not exceed the theoretical amount of energy it can eventually release, these values can be used to indicate the 'order of magnitude' of the absolute lowest grade of ore that can be processed by conventional methods.* Using this

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* This assumes that all other energy inputs to the system are insignificant compared to E_u. See following sections, however.

criterion, figure 5.2 shows that the approximate limit to the grade of ore that can be used by a burner reactor system varies between 1 and 60 parts per million U_3O_8 (10^{-6}: ppm U_3O_8) with a mean value of 10 ppm U_3O_8. Limits for breeder reactor systems range from 7 to 500 parts per billion U_3O_8 (10^{-9}: ppb U_3O_8) with a mean of 70 ppb U_3O_8.

These approximate results suggest that the amount of uranium available from ores which can be processed by conventional means is restricted by energy considerations. Results of a resource evaluation given in appendix F indicate world resources of orthodox ores of the order of 10^9 tonnes U_3O_8 for burner reactor systems and 10^{11} tonnes U_3O_8 for breeders. It should be noted, however, that these results are only approximations to the actual ore grade limits and subsequently viable resource bases since energy inputs to the remaining stages of the fuel cycle, and to reactor construction and operation are excluded from this simple preliminary assessment. More reliable results can obviously be obtained from a detailed, more complete analysis (see chapters 8 and 9).

In addition to the conventional techniques examined so far, there are other methods which can be used to produce uranium concentrate from resources. New processes such as 'in situ' leaching are being developed to treat currently-commercial ores (see Mining Magazine, 1971b) and numerous techniques have been proposed for the treatment of uranium-bearing phosphoric acid (Clegg and Foley, 1958), copper leach solutions (Bieniewski et al, 1971) and seawater (Davies et al, 1964).

In situ leaching, which consists of pumping reagent through a deposit to extract uranium directly from the ore, is an attractive method since expensive mining and comminution operations can be avoided. This reduces the energy input to uranium concentrate production, in some cases by almost 50% (see appendix D), but wide scale application of this technique is restricted by demanding geological requirements and low recovery efficiency rates.

Uranium contained in phosphate rocks represents a fairly substantial medium-grade (50 - 200 ppm U_3O_8) resource of nuclear fuel. Uranium can be recovered from phosphoric acid produced with uraniferous phosphates by the 'wet process' technique (Bieniewski et al, 1971). Assuming that uranium obtained by this method is treated as a by-product, then only the energy specifically involved in recovery operations is included in the resulting energy requirement. Although such a by-product process requires less energy than the conventional processing of orthodox ores (see appendix D), uranium production rates are determined by the demand for phosphates produced by this method rather than the demand for uranium. Hence this is a secondary route to uranium concentrate and, as such, it is unlikely to be a major source of nuclear fuel.

The copper leach solution process is another secondary technique for uranium production. This involves extracting uranium from leach liquors which have been used to treat uraniferous copper ores. By-product uranium concentrate obtained in this manner only incurs the energy inputs of recovery operations and therefore has a lower energy requirement than that of yellow cake obtained by conventional means from orthodox ores (see

appendix D). However, the copper leaching process is only used with a small proportion of cupriferous ores and consequently this method is not expected to contribute significantly to nuclear fuel supply.

Since seawater contains 3.3 ppb uranium in solution, it has often been regarded as a potentially important source of nuclear fuel. Numerous extraction techniques and treatment schemes have been proposed, but only certain methods seem to be practical in both financial and energy terms (see appendix E). The most attractive schemes appear to be those that would use a titanium hydroxide absorption cycle. This involves the selective absorption of uranium on titanium hydroxide granules, followed by re-dissolution and uranium recovery by direct solvent extraction (D.S.E.) using salt as an extractant or by steam stripping (S.S.) which reduces salt consumption.

Extraction projects using this cycle must achieve a fairly high throughput of uranium-bearing seawater to be commercially viable. On this basis tidal and combined treatment schemes appear to be the most feasible. Tidal schemes would attain high production rates by impounding large areas of water which were continually replenished by the flow of the tides. By combining uranium extraction units with other schemes that pump vast quantities of seawater for desalination and cooling systems, similar rates of output could be achieved. Although the fuel consumption of such combined schemes would be high, the energy requirement of uranium production could be unaffected if the energy input to pumping was totally transferred to main products (eg. fresh water in the case of desalination plants and electricity from coastal power stations; see appendix E).

Despite reasonably moderate energy requirements, however, initial investigation indicates that seawater treatment schemes may only be able to supply limited amounts of nuclear fuel. A shortage of suitable sites appears to restrict the production of uranium from tidal schemes to about 10^4 tonnes U_3O_8 per year and the practical commercial size of desalination plants, coastal power stations, etc., limits the widespread use of combined schemes. Uranium produced by these methods is also likely to be expensive - at least ten times present costs (Harrington et al, 1974).

Further details of all these alternative methods of producing uranium concentrate are given in appendices D and E. Results of energy analysis are summarised in figure 5.3 and the current potential contribution of each process to nuclear fuel supply is illustrated in figure 5.4. Production rates for the by-product wet-process phosphoric acid and copper leach solution processes are determined by the relative demand for wet-process phosphates and copper leach concentrates respectively (eg. U. S. Bureau of Mines, 1972; von Kienlin, 1976). The potential contribution of seawater extraction schemes is fixed by the number of suitable sites (see appendix E). The results of figure 5.4 can be compared with current world production rates by conventional means from orthodox ores of 2×10^4 tonnes U_3O_8 per year and a forecast requirement of about 3×10^5 tonnes annually by 2000 (Nuclear Energy Agency, 1975).

The effect of using different sources of uranium on the energy requirement of uranium concentrate was deduced from figure 5.5 which shows the estimated variation of world uranium resources

Figure 5.3 : Energy requirements of alternative methods of producing uranium concentrate (assuming 4 MJ[t]/MJ[e])

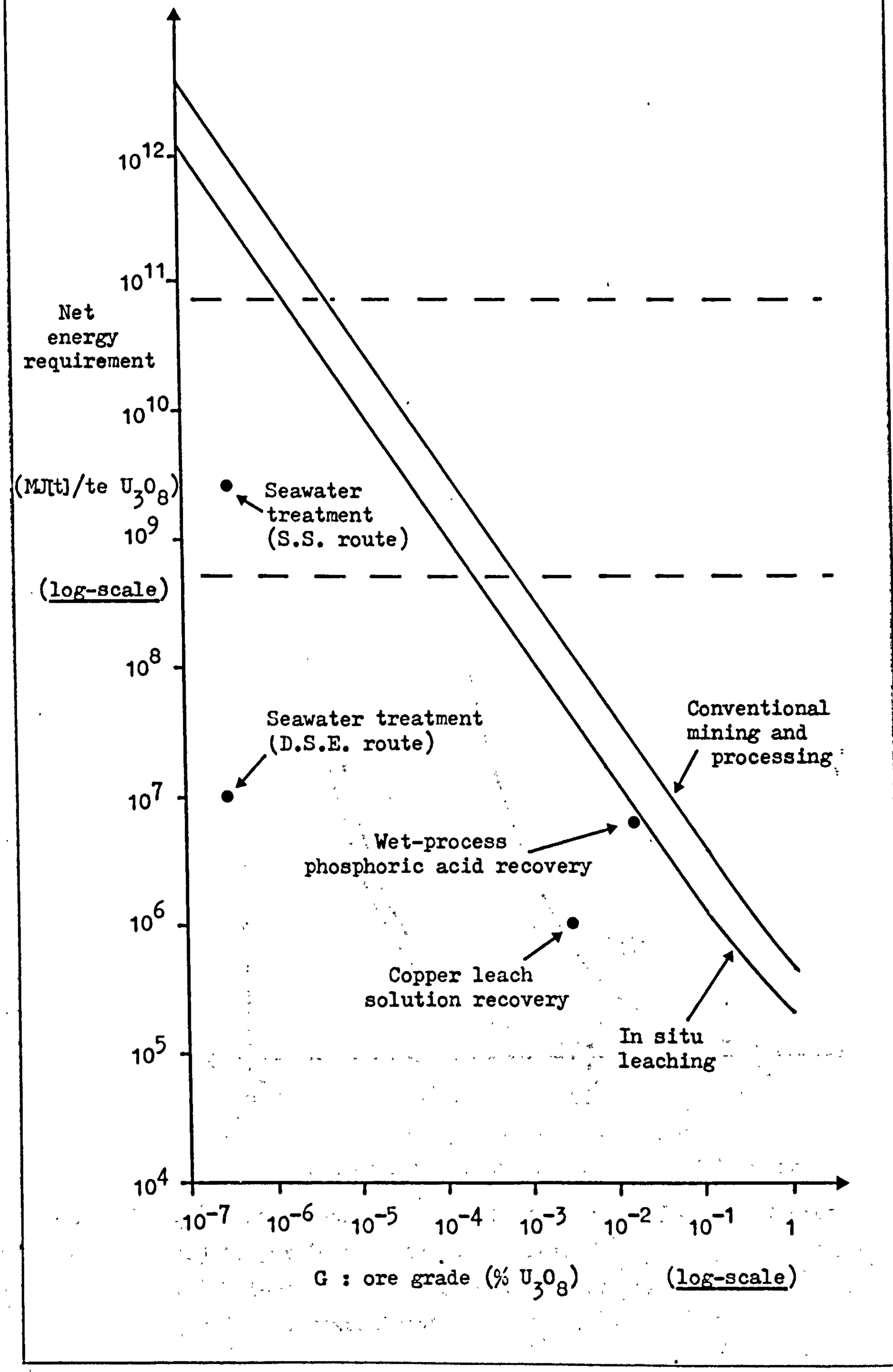
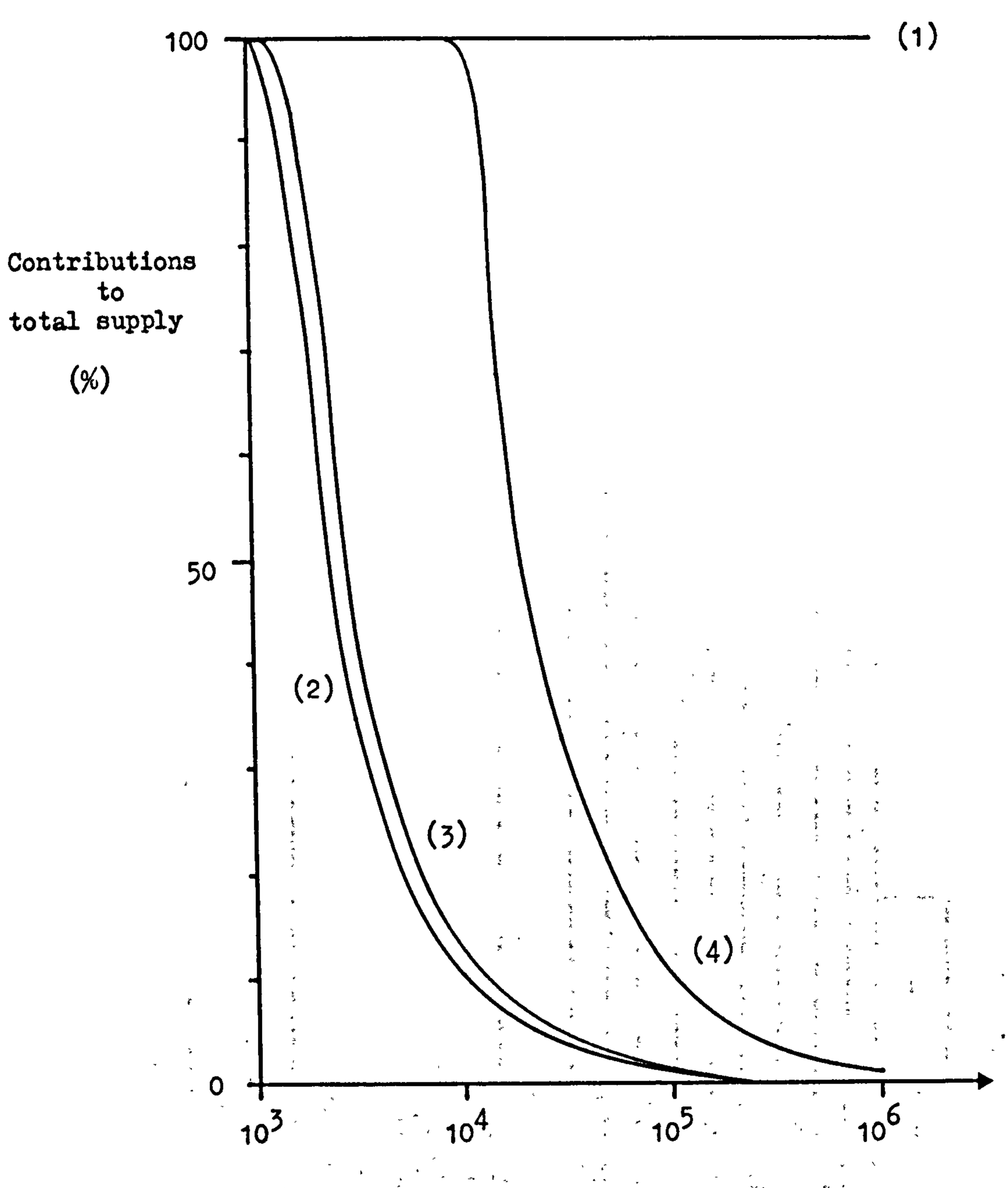
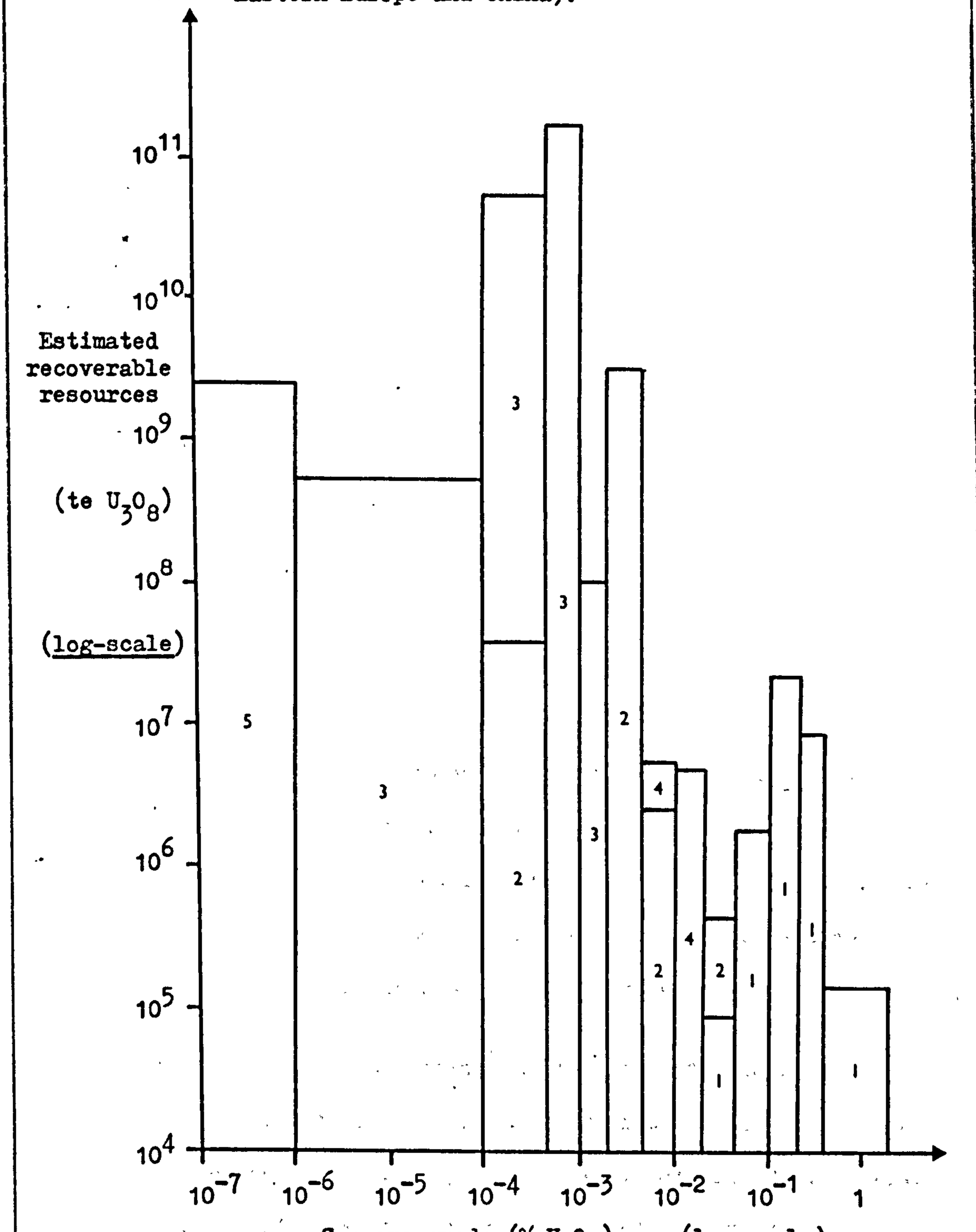


Figure 5.4 : Potential contributions of different production methods to the supply of uranium concentrate.



Key: (1) Conventional mining and processing of orthodox ores
(2) Copper leach solution recovery
(3) Wet-process phosphoric acid recovery
(4) Seawater treatment

Figure 5.5 : <u>A speculative assessment of ultimate world uranium resources</u> (excluding the resources of the USSR, Eastern Europe and China).



Key: (1) Common commercial ores
(2) Carbonaceous material
(3) Other primary sources
(4) Phosphates
(5) Seawater

with ore grade. This postulated resource profile, which was evaluated from the same data used to assess the results of appendix F, indicates the ultimate amount of uranium that may exist, in various forms, throughout the world. This speculative assessment was used to calculate the variation of the uranium concentrate energy requirement with cumulative resources and is shown in figure 5.6 assuming that sources of uranium are exploited in order of increasing energy requirement. By comparison with the energy available from U-235 fission in one tonne of triuranium octoxide, these results indicate that cumulative resources cannot exceed about 10^{10} tonnes U_3O_8 on the basis of the energy input to uranium concentrate production alone.

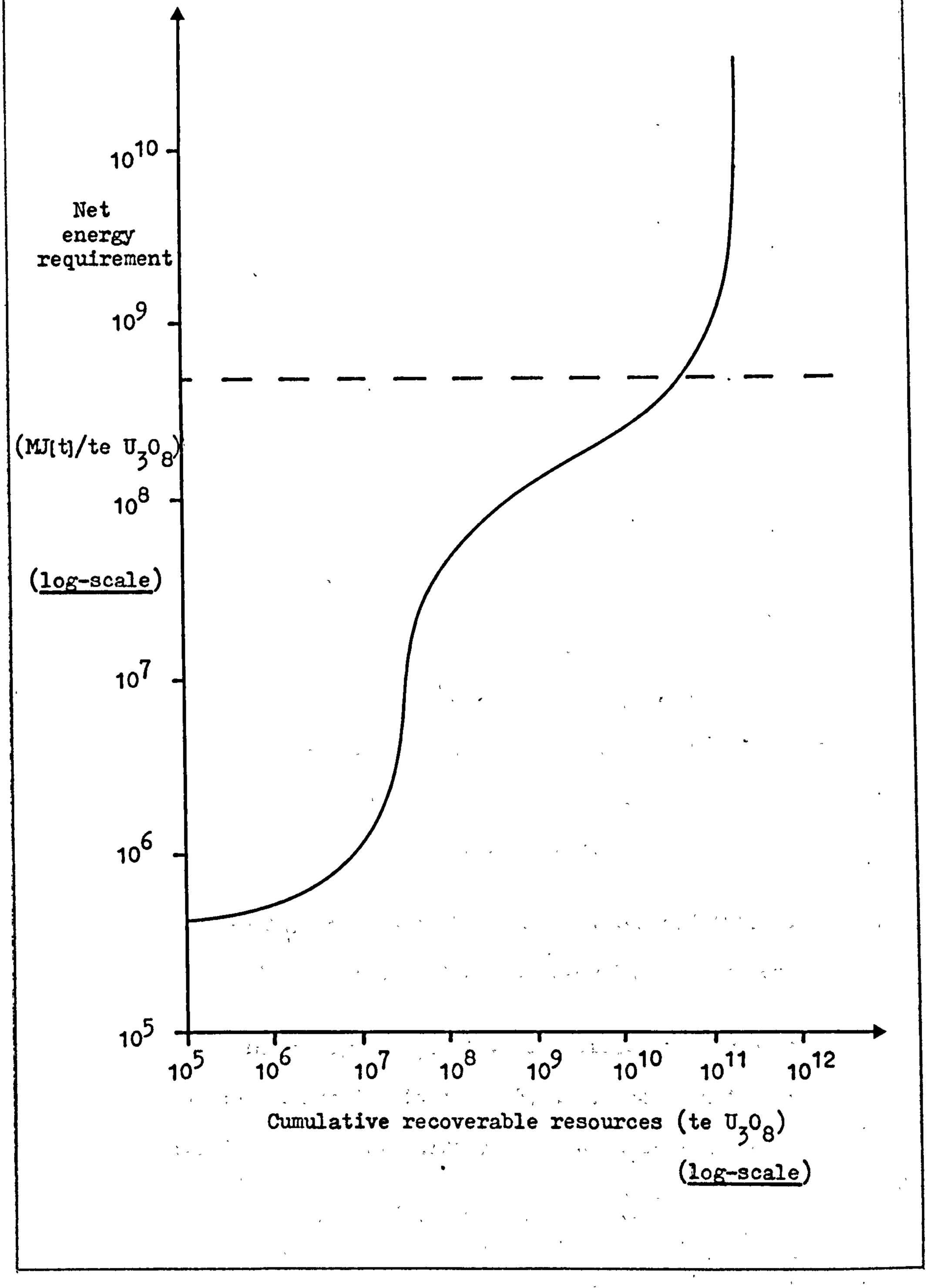
5.3 Production of fuel from uranium concentrate

5.3.1 Refining and conversion

Uranium concentrate produced from ore contains impurities that would interfere with nuclear reactions. Additionally, concentrates are chemically and physically unsuitable for use in reactor fuel and must be converted to more convenient compounds at this stage of the fuel cycle.

Refining consists of dissolving the concentrates in acid and selectively extracting uranium from solution with appropriate organic reagents. Very pure uranyl nitrate is produced which is then denitrated to obtain uranium dioxide powder. Subsequent conversion operations are determined by the type of reactor fuel being prepared. The six different reactor fuel cycles examined in this study include those which supply the MAGNOX, Advanced Gas-cooled Reactor (AGR), Steam Generating

Figure 5.6 : <u>Estimated variation of the aggregated average energy requirement of uranium concentrate with cumulative resources</u> (assuming an electrical conversion factor of 4 MJ[t] per MJ[e])



Heavy Water Reactor (SGHWR), CANDU, Boiling Water Reactor (BWR) and Pressurised Water Reactor (PWR) designs.

The CANDU reactor uses natural, or unenriched, uranium dioxide fuel which contains the natural abundance of 0.71% of the fissile isotope uranium-235 (U-235). Hence no further chemical processing of CANDU fuel cycle material is required after refining at this stage. The MAGNOX reactor also consumes natural uranium but in metallic form. Consequently uranium dioxide produced during refining must be reduced to uranium metal for MAGNOX fuel use.

All remaining reactor types operate on enriched uranium dioxide fuel which contains an artificially increased proportion of U-235. Since most common uranium isotope enrichment techniques use uranium in gaseous form, uranium dioxide obtained from refining must be converted to uranium hexafluoride gas, or 'hex', at this point.

Details of all refining and conversion procedures are given in appendix G which also indicates the energy requirements of these various refining and conversion processes. The energy required by the refining and conversion of uranium concentrate can be summarised by the following term;

E_4 = energy input per unit mass uranium refined, converted and delivered to the next step in the fuel cycle

Values of the energy input, E_4, for different reactor fuel cycles are illustrated in table 5.9. These results were deduced from the information presented in appendix G.

Table 5.9 : Energy input to the refining and conversion of uranium concentrate.

Reactor fuel cycle	Energy input : E_4 (10^6 MJ/te U)	
	minimum	maximum
MAGNOX :	0.061(e) + 0.067(t)	0.061(e) + 0.316(t)
AGR :	0.043(e) + 0.153(t)	0.050(e) + 0.310(t)
SGHWR :	0.043(e) + 0.153(t)	0.050(e) + 0.310(t)
CANDU :	0.033(e) + 0.031(t)	0.034(e) + 0.151(t)
BWR :	0.043(e) + 0.153(t)	0.050(e) + 0.310(t)
PWR :	0.043(e) + 0.153(t)	0.050(e) + 0.310(t)

5.3.2 Isotope enrichment

Uranium isotope enrichment which consists of enhancing the relative abundance of the fissile isotope U-235 is an important step in the fuel cycle of certain burner reactor designs. Isotopes of the same element have the same atomic number, which determines chemical properties, but different mass numbers, which cause differences in physical properties. Hence the separation of isotopes must be primarily based on physical characteristics which depend on relative isotopic mass differences. For a heavy element such as uranium the relative mass difference between the natural isotopes U-235 and U-238 is small (in fractional terms, about 0.013) and consequently enrichment techniques must be very sensitive.

Because uranium enrichment relies on such slight differences in physical properties, isotopic separation consumes substantial quantities of energy. Although the theoretical amount of energy required to totally separate U-235 and U-238 is approximately 200 MJ per tonne of uranium, in practice the direct energy requirement of actual processes can be as much as ten million times greater (McGeoch, 1977). The total energy input to this stage of the fuel cycle can be represented by the following relationship;

E_5 = energy input per unit mass uranium enriched

$= e_{5a} \times s$

where,

e_{5a} = e.r. of uranium enrichment per unit mass separative work

s = units mass of separative work required per unit mass of enriched uranium produced

= separative work requirement

The e.r. of enrichment, e_{5a}, depends on the type of separation technology in use. A number of different enrichment processes are currently available or under development and these are described in appendix H. Some results of the energy analysis of enrichment methods presented in this appendix are shown in table 5.10. This illustrates e.r. values for the widely-used gas diffusion process and the newly-commercialised gas centrifuge method.

The gas diffusion process consists of a large number of porous membranes through which uranium hexafluoride gas is pumped. Partial separation is achieved at each membrane by virtue of the different rates of diffusion of gas containing U-235 and U-238 atoms. The gas centrifuge method is based on differences in the centrifugal forces acting on gas molecules in a rotating cylinder. Marginally higher concentrations of gas containing the lighter isotope collect near the axis of such vessels and many centrifuges are required to obtain suitably enriched uranium.

Although the gas centrifuge technique uses considerably less energy than the gas diffusion method (see table 5.10), new processes which could achieve still low fuel consumption are currently being developed (see appendix H). However, these processes may take many years to commercialise and it is unlikely that they will supply much enriched fuel over the next decade.

Table 5.10 : Estimated energy requirement for uranium isotope enrichment.

Process	Energy requirement : e_{5a} (10^6 MJ/te S.W.U.)	
	minimum	maximum
Gas diffusion :	8.47(e) + 0.13(t)	11.27(e) + 0.44(t)
Gas centrifuge :	0.87(e) + 0.17(t)	1.16(e) + 0.82(t)

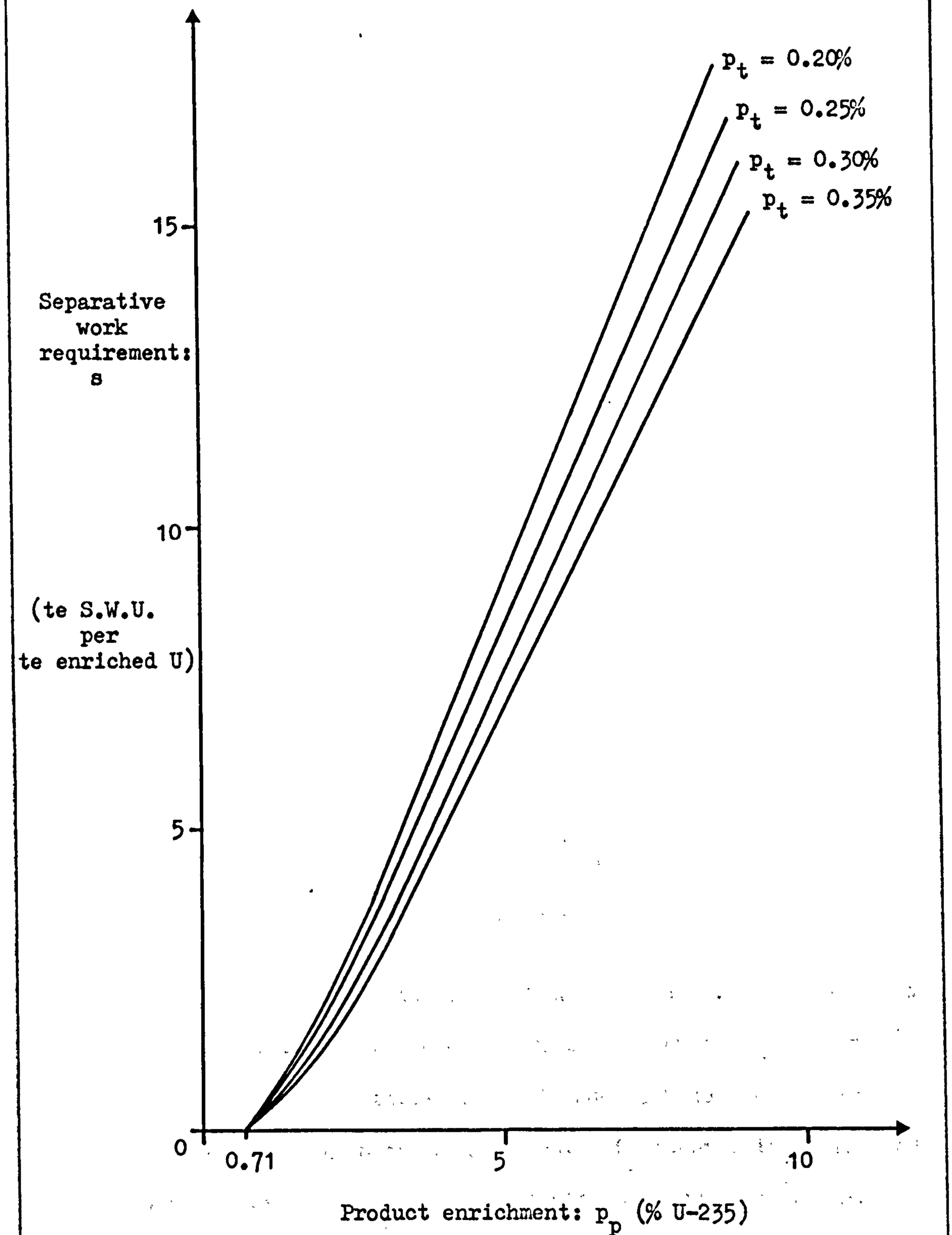
The e.r. of enrichment, e_{5a}, measured in terms of energy per unit mass of separative work (MJ/te S.W.U.), is independent of the level of enrichment achieved. To deduce the energy input per unit mass of uranium enriched to a given percentage of U-235, it is necessary to multiply e_{5a} by the separative work requirement, s. Separative work is a measure of the amount of enrichment required to produce enriched material. This is a function of the percentage of U-235 in the initial uranium, p_f, the enrichment percentage of the final product, p_p, and the percentage of U-235 in the waste uranium, or tails, created by the separation process, p_t. The variation of the separative work requirement, s, with the desired level of enrichment, p_p, using natural uranium as starting material (p_f = 0.71% U-235) is shown in figure 5.7 for different values of the tails assay, p_t. This diagram incorporates data from standard literature on enrichment principles (Bader, Kitzke and Norman, 1969; Avery and Kehoe, 1970; Eister and Kennedy, 1974).

In addition to deciding the value of the separative work requirement, the factors p_f, p_p and p_t also determine the amount of uranium required to produce enriched uranium. Assuming the conservation of mass and the conservation of U-235 atoms, the quantity of uranium, containing p_f% U-235, fed into a separation process, which produces p_t% U-235 tails and p_p% U-235 enriched uranium output, can be written as;

f = amount of feed uranium required per unit mass enriched uranium output

$$= \frac{(p_p - p_t)}{(p_f - p_t)} .$$

Figure 5.7 : Separative work requirement for the isotope enrichment of natural uranium.



The variation of this formula for natural uranium feed (p_f = 0.71% U-235) with p_p and different fixed values of p_t is illustrated in figure 5.8. The feed requirement, f, is an important factor in the evaluation of the net energy required to produce enriched fuel elements from ore (see 5.5.1 and 5.5.2).

5.3.3 Fuel fabrication

Fuel fabrication consists of forming natural and enriched uranium compounds into fuel elements which can be used to provide energy in reactors. This involves converting and shaping nuclear material from refineries or enrichment plants into fuel slugs or pellets that can be made into fuel rods. These rods are covered in a protective material, or cladding, and then assembled into arrays called fuel elements which can be placed in the core of the reactor.

The total energy input to this stage of the fuel cycle can be represented by the following single term;

E_6 = energy input per unit mass uranium fabricated into fuel elements and delivered to the reactor

The energy requirements of fuel fabrication for various types of reactor fuel cycle are examined in appendix I and results are summarised in table 5.11. In addition to fuel element manufacture the energy requirements of transporting fuel from the fabrication plant to the reactor are included in these results.

The processes involved in fabrication and subsequent energy inputs are determined by the sort of reactor fuel being

Figure 5.8 : Variation of the natural uranium feed input to isotope enrichment with product enrichment.

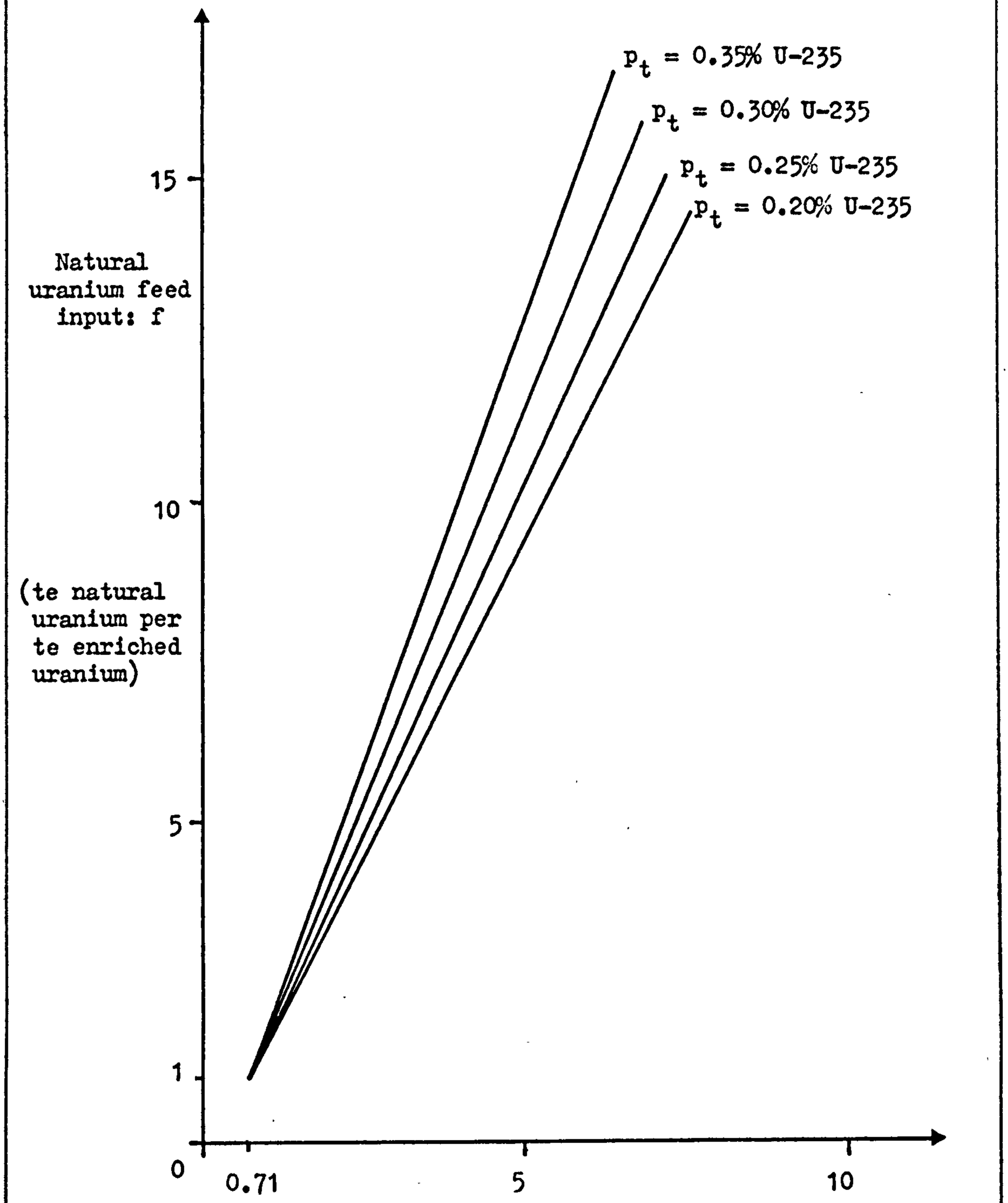


Table 5.11 : Estimated energy inputs to the fabrication of uranium fuel.

Reactor fuel cycle	Energy input : E_6 (10^6 MJ/te U) minimum	maximum
MAGNOX :	0.145(e) + 0.102(t)	0.154(e) + 0.167(t)
AGR :	0.179(e) + 0.238(t)	0.190(e) + 0.283(t)
SGHWR :	0.200(e) + 0.260(t)	0.223(e) + 0.334(t)
CANDU :	0.171(e) + 0.129(t)	0.196(e) + 0.218(t)
BWR :	0.213(e) + 0.274(t)	0.243(e) + 0.360(t)
PWR :	0.215(e) + 0.277(t)	0.246(e) + 0.365(t)

produced. Fuel elements for the MAGNOX reactor design are simply made by machining natural uranium metal obtained from the refinery into rods and encasing them in a magnesium alloy known as magnox. The manufacture of CANDU fuel elements consists of forming pellets from natural uranium dioxide powder and incorporating them into rods sheathed in a zirconium alloy called zircalloy-4. The starting material for fuel for reactors such as the AGR, SGHWR, BWR and PWR that use enriched uranium is uranium hexafluoride gas from the enrichment plant. This must be converted to uranium dioxide which can be pelletised and clad with zirconium alloys such as zircalloy-2 and zircalloy-4. Further details of these operations are given in appendix I.

5.4 Recovery of fissile-bearing material from spent uranium fuel

5.4.1 Reprocessing

Uranium fuel exposed to neutron irradiation in the reactor core produces heat by fission and a series of isotopes called fission products. These isotopes can seriously affect the nuclear, thermal and mechanical properties of the fuel and consequently it is necessary to periodically remove fuel from the reactor to extract these products. This is achieved by reprocessing which enables fissile-bearing materials, such as unreacted uranium and plutonium created from conversion reactions between uranium and neutrons, to be recovered from the irradiated, or 'spent' fuel. Following separation and further processing, uranium and plutonium can be incorporated into new fuel elements and re-introduced to the reactor. Fission products are usually treated as waste materials.

Typical reprocessing operations are described in appendix J. These consist of removing the irradiated material from spent fuel rods and dissolving it in acid. Uranium and plutonium are recovered from the solution containing fission products by a selective organic solvent-extractant. The fission products in solution form liquid effluents which can be disposed of by waste management schemes. The recovered uranium and plutonium are separated and purified to obtain uranium and plutonium nitrate. Uranium nitrate can then either be converted to uranium dioxide for direct re-introduction to nuclear fuel or converted to uranium hexafluoride for re-enrichment prior to re-cycling. Plutonium nitrate is usually converted to plutonium dioxide for use in either burner or breeder reactors.

The energy required to reprocess spent fuel can be represented by the following sum of energy requirements;

E_7 = energy input per unit mass fissile-bearing material (U or Pu) recovered from spent uranium fuel and prepared for re-introduction into new fuel elements

$$= e_{7a} + e_{7b}$$

where,

e_{7a} = e.r. of recovering uranium and plutonium nitrate from spent fuel

e_{7b} = e.r. of converting uranium and plutonium nitrate for re-cycling

The e.r. of uranium and plutonium recovery, e_{7a}, is evaluated in appendix J and results are summarised in table 5.12. Energy inputs to all transportation required during reprocessing are included in these figures. The e.r. of nitrate conversion, e_{7b},

Table 5.12 : Estimated energy requirement of recovering uranium and plutonium nitrate from spent fuel.

Process	Energy requirement: e_{7a} (10^6MJ/te fissile-bearing material)	
	minimum	maximum
PUREX :	0.047(e) + 0.087(t)	0.090(e) + 0.199(t)

is assessed in appendix G and results are given in table 5.13. The total energy input to reprocessing and re-cycling, E_7, is illustrated in table 5.14.

5.4.2 Waste disposal

Waste liquids and solids from nuclear fuel reprocessing can be divided into three general categories depending on how many fission products they contain and how much radioactivity they subsequently emit. High-level wastes, which contain the highest concentration of fission products as well as traces of uranium and plutonium, consist of reprocessing liquid effluents and solid waste cladding material. Intermediate-level wastes are also waste solutions from reprocessing but contain fewer fission products. The lowest concentration of fission products occurs in low-level wastes which include cleaning solvents and various contaminated solids.

These categories of waste can be treated in numerous different ways. Although many schemes have been proposed for the disposal of high-level wastes, one particular type of method is currently attracting much interest. This consists of concentrating the waste liquids by evaporation, storing them in cooled tanks for a few years to allow certain radioactive isotopes to decay and then solidifying the waste into glass blocks or calcined ash which can be encased in metallic containers that could be disposed underground or beneath the sea. Intermediate-level wastes which are generally less radioactive can be either discharged into the sea after a short period of storage or stored permanently in underground tanks. Low-level liquid wastes are usually released directly into the environment, whilst solids are buried in pits.

Table 5.13 : Estimated energy requirements for the conversion of uranium and plutonium nitrate from fuel reprocessing.

Final product	Energy requirement : e_{7b} (10^6 MJ/te fissile-bearing material)	
	minimum	maximum
Uranium and plutonium dioxide :	0.008(t)	0.108(t)
Uranium hexafluoride :	0.011(e) + 0.130(t)	0.015(e) + 0.255(t)

Table 5.14 : Estimated energy input to the reprocessing and re-cycling of uranium and plutonium from spent fuel.

Product form	Energy input: E_7 (10^6 MJ/te fissile-bearing material)	
	minimum	maximum
Uranium and plutonium dioxide :	0.047(e) + 0.095(t)	0.090(e) + 0.307(t)
Uranium hexafluoride :	0.058(e) + 0.217(t)	0.105(e) + 0.454(t)

Further details of waste management schemes and the energy analysis of these operations are presented in appendix K. The energy contribution of waste disposal to the reprocessing of nuclear fuel can be summarised by the term;

E_8 = energy input to waste management per unit mass fissile-bearing material (U or Pu) reprocessed

The energy input, E_8, of the currently most popular waste management proposals are shown in table 5.15.

5.5 Net energy requirements of nuclear fuel

The net energy requirement of nuclear fuel consists of the sum of all energy inputs to the fuel cycle excluding the energy content of the initial material from which the fuel is made. The type of reactor for which the fuel is manufactured determines its chemical and physical composition, hence the particular process routes used in the fuel cycle and consequently the net energy requirement of the fuel. Additionally, the original source of nuclear material from which fuel is obtained can decide the techniques and therefore the energy used in the fuel cycle. Material produced from natural uranium resources such as ores is referred to here as 'primary-produced fuel' and material containing uranium and/or plutonium recovered by reprocessing spent nuclear fuel is called 'secondary-produced fuel'. The net energy requirements of both types of fuel for the six main burner reactor designs are evaluated in this section.

Table 5.15 : Estimated energy input contribution to fuel reprocessing of nuclear waste management.

Value	Energy input : E_8 (10^6 MJ/te fissile-bearing material)
Minimum :	0.019(e) + 0.004(t)
Maximum :	0.055(e) + 0.052(t)

5.5.1 Primary-produced fuel

Primary-produced fuel is manufactured entirely from naturally-occurring sources of uranium and contains no re-cycled nuclear material obtained from the reprocessing of spent fuel that has previously been irradiated in a reactor. The fuel cycle used to produce such fuel consists of exploration, uranium ore mining and processing, uranium concentrate refining and conversion, isotope enrichment as required and fuel fabrication. The total amount of energy required by this fuel cycle can be represented by the following expression;

N_A = net energy requirement of primary-produced fuel

$$= \left[\left(\frac{E_1}{r_2 \times r_3} + \frac{E_2}{r_3} + E_3 \right) \times 1.18 + E_4 \right] \times f + E_5 + E_6$$

where,

E_1 = energy input per unit mass U_3O_8 discovered

E_2 = energy input per unit mass U_3O_8 mined and sent to mill

E_3 = energy input per unit mass U_3O_8 processed and sent to refinery

E_4 = energy input per unit mass uranium refined, converted and delivered to the next step in the fuel cycle

E_5 = energy input per unit mass uranium enriched

E_6 = energy input per unit mass uranium fabricated into fuel elements and delivered to the reactor

r_2 = amount of U_3O_8 recovered by mining per amount of U_3O_8 in the deposit

r_3 = amount of U_3O_8 produced in concentrate per amount of U_3O_8 processed in ore

f = natural uranium feed per unit mass uranium enriched

Estimates of the values of these particular energy inputs are available from previous sections of this chapter. The energy inputs to exploration, E_1, mining, E_2, and ore processing, E_3, combine with the mining and processing recovery efficiency factors, r_2 and r_3 respectively, shown in table 5.7, to produce the estimates of the net energy requirement of uranium concentrate, E_u, given in table 5.8. This n.e.r. depends on the fraction of triuranium octoxide (U_3O_8) in the ore, that is the ore grade, G, and is measured in terms of energy per unit mass U_3O_8. The n.e.r. is converted to energy per unit mass uranium by multiplying E_u by a factor of 1.18, the amount of U_3O_8 per unit mass U.

All terms in the equation for the net energy requirement of primary-produced fuel, N_A, are influenced, either directly or indirectly, by the type of reactor being supplied. The term for the energy input to refining and conversion, E_4, depends on the process routes used and this is affected by the type of reactor fuel cycle in operation (see table 5.9). Additionally, the energy input to the production of a refined basic uranium product from ore, which is the sum of E_u and E_4, is influenced by the amount of natural uranium required to obtain a unit mass of enriched uranium, f. This is determined by the level of enrichment of the fuel used in the reactor, p_p, and the percentage of uranium-235 rejected during enrichment, p_t. The variation of f with these parameters is illustrated in figure 5.8.

The energy input to enrichment, E_5, depends on the type of enrichment technique used and it is also influenced by reactor design through the separative work requirement, s (see 5.3.2).

The factor s indicates the amount of enrichment required to produce uranium containing p_p% U-235 and the variation of s with p_p and p_t is shown in figure 5.7. The energy input to fuel fabrication, E_6, depends on the final chemical composition of the fuel and the type of material used for cladding. Hence E_6 is directly determined by reactor design as illustrated in table 5.11.

It will be noted that the energy inputs to reprocessing, E_7, and waste management, E_8, are excluded from the equation for the n.e.r. of primary-produced fuel. Although reprocessing and waste management are part of the fuel cycle, spent nuclear fuel is frequently left untreated and stored in cooling ponds at the moment due to shortages of reprocessing capacity and the general lack of incentive to re-cycle (Vielvoye, 1976). Any fissile-bearing material that is recovered from spent fuel can be incorporated into re-cycled, or secondary-produced fuel, consumed by reactors other than burner designs, or used for military purposes. Consequently the energy inputs E_7 and E_8 were attributed to these products rather than primary-produced fuel.

To evaluate the n.e.r. of fuel for different reactor types it is necessary to specify particular parameters such as the fuel enrichment, p_p, the natural uranium feed to enrichment, f, and the separative work requirement, s. These parameters for the six main burner reactor designs studied here were deduced from typical operating data (International Atomic Energy Agency, 1967, 1970a, 1972; Nuclear Engineering International, 1972) and the information given in figures 5.7 and 5.8. Values of these parameters for the fuel used to build the initial cores of

reactors and that included in replacement loadings, or reloads, during operation are shown in table 5.16. A typical tails assay for enrichment, p_t, of 0.25% U-235 was assumed in these calculations.

By combining this information with results obtained earlier in this chapter, estimates of the n.e.r. of primary-produced fuel, N_A, were deduced. These estimates are illustrated in table 5.17. Most results were evaluated assuming any necessary uranium isotope enrichment was achieved with the gas diffusion technique (see 5.3.2). For comparison, values of N_A for the PWR design were also calculated assuming gas centrifuge enrichment.

5.5.2 Secondary-produced fuel

Secondary-produced fuel is re-cycled nuclear material obtained by reprocessing spent primary-produced fuel that has been irradiated and then removed from the reactor. Re-cycled fuel can consist of reprocessed, depleted and re-enriched uranium or plutonium. Spent fuel initially contains depleted uranium which has a smaller proportion of U-235, p_d, than the original fraction before irradiation, p_p. Additionally, spent fuel contains plutonium created from U-238 by conversion reactions with neutrons. Reprocessing recovers this fissile-bearing material as uranium and plutonium nitrate. Uranium nitrate can either be converted directly into uranium dioxide powder or turned into uranium hexafluoride gas which can be re-enriched. Depleted uranium dioxide can be fabricated into fuel rods for breeder reactors, whilst re-enriched uranium hexafluoride can

Table 5.16 : Fuel enrichment parameters for typical reactor designs.

Reactor	Fuel enrichment : p_p* (% U-235)	Natural uranium feed : f† (te natural U per te enriched U)	Separative work : s† (te S.W.U./ te enriched U)
MAGNOX			
- initial core and replacement loadings :	0.71	1.00	0
AGR			
- initial core :	1.65	3.05	1.35
- replacement loadings :	2.25	4.35	2.30
SGHWR			
- initial core and replacement loadings :	2.10	4.00	2.10
CANDU			
- initial core and replacement loadings :	0.71	1.00	0
BWR			
- initial core :	2.25	4.30	2.30
- replacement loadings :	2.60	5.05	3.00
PWR			
- initial core :	2.50	4.85	2.80
- replacement loadings :	3.10	6.15	4.00

* weighted average value for zoned cores

† assumes a typical tails assay of 0.25% U-235

Table 5.17 : Net energy requirements for the primary-produced fuel of various burner reactor designs.

Reactor	Net energy requirement : N_A^* (10^6 MJ/te U)			
	minimum		maximum	
	electrical	thermal	electrical	thermal
MAGNOX ; initial core & reload fuel	: $\frac{(0.006+0.21)}{G} + \frac{(0.020+0.17)}{G}$		$\frac{(0.85+0.22)}{G} + \frac{(1.1+4.1)}{G}$	
AGR ; initial core	: $\frac{(0.019+11.8)}{G} + \frac{(0.062+0.88)}{G}$		$\frac{(2.59+15.6)}{G} + \frac{(3.4+13)}{G}$	
reload fuel	: $\frac{(0.027+19.8)}{G} + \frac{(0.090+1.20)}{G}$		$\frac{(3.69+26.3)}{G} + \frac{(4.8+18)}{G}$	
SGHWR ; initial core & reload fuel	: $\frac{(0.025+18.2)}{G} + \frac{(0.081+1.14)}{G}$		$\frac{(3.39+24.1)}{G} + \frac{(4.4+17)}{G}$	
CANDU ; initial core & reload fuel	: $\frac{(0.006+0.20)}{G} + \frac{(0.020+0.16)}{G}$		$\frac{(0.85+0.23)}{G} + \frac{(1.1+4.0)}{G}$	
BWR ; initial core	: $\frac{(0.027+19.9)}{G} + \frac{(0.088+1.23)}{G}$		$\frac{(3.65+26.4)}{G} + \frac{(4.8+18)}{G}$	
reload fuel	: $\frac{(0.032+25.8)}{G} + \frac{(0.103+1.44)}{G}$		$\frac{(4.28+34.3)}{G} + \frac{(5.6+22)}{G}$	
PWR ; initial core	: $\frac{(0.030+24.1)}{G} + \frac{(0.099+1.38)}{G}$		$\frac{(4.11+32.0)}{G} + \frac{(5.4+21)}{G}$	
reload fuel	: $\frac{(0.039+34.4)}{G} + \frac{(0.125+1.74)}{G}$		$\frac{(5.22+45.6)}{G} + \frac{(6.8+26)}{G}$	
PWR† ; initial core	: $\frac{(0.030+2.30)}{G} + \frac{(0.099+1.49)}{G}$		$\frac{(4.11+3.74)}{G} + \frac{(5.4+22)}{G}$	
reload fuel	: $\frac{(0.039+3.16)}{G} + \frac{(0.125+1.90)}{G}$		$\frac{(5.22+5.20)}{G} + \frac{(6.8+28)}{G}$	

* assumes gas diffusion enrichment except where indicated

† assumes gas centrifuge enrichment

be converted into uranium dioxide and made into fuel rods for burner reactors. Plutonium dioxide powder can be obtained from plutonium nitrate and mixed with re-enriched uranium dioxide for burner reactor fuel or used separately in breeder reactor fuel.

The fuel cycle for re-cycled, re-enriched uranium fuel consists of reprocessing with subsequent waste disposal, followed by re-enrichment and fuel fabrication. Hence the net energy requirement can be represented by the expression;

N_B = net energy requirement of re-enriched secondary-produced uranium fuel

$$= E_5' + E_6 + (E_7 + E_8) \times f'$$

where,

E_5' = energy input per unit mass uranium re-enriched

E_6 = energy input per unit mass uranium fabricated into fuel elements and delivered to the reactor

E_7 = energy input per unit mass uranium recovered from spent fuel by reprocessing and prepared for re-cycling

E_8 = energy input to waste management per unit mass uranium reprocessed

f' = amount of irradiated uranium feed per unit mass re-enriched uranium

The energy inputs to reprocessing, E_7, and waste management, E_8, are independent of the specific reactor design, but their combined contribution to the n.e.r. of secondary-produced fuel, N_B, is influenced by the amount of reprocessed, depleted uranium required to produce a unit mass of re-enriched fuel, f', which is a function of the reactor enrichment parameters. The energy input to re-enrichment, E_5', is also determined by these

parameters through the separative work requirement, s'. The energy input to fuel fabrication, E_6, is directly affected by reactor type (see table 5.11).

The parameters which determine the re-enrichment feed requirement factor, f', and the separative work requirement, s', are the fraction of U-235 in spent or depleted fuel, p_d, the fraction of U-235 in re-enriched fuel, p'_p, and the fraction of U-235 in the enrichment tails, p_t. The enrichment of re-cycled fuel, p'_p, does not simply equal the enrichment of initial primary-produced fuel, p_p, since uranium which has been irradiated contains the artificially produced isotope U-236. It has been suggested (Sprague, 1974; Resnikoff, 1975) that since this isotope is a neutron absorber, it is necessary to enrich re-cycled uranium above the normal level to counteract the undesireable nuclear properties of this isotope. The new enrichment, p'_p, which displays roughly the same fission characteristics as p_p% U-235 enriched primary-produced fuel, depends on the amount of U-236 in the depleted uranium. The U-236 content of depleted uranium from reactors operating under typical conditions averages 0.5% U-236 and the new level of re-enrichment is given by the equation;

$$p'_p = (p_p + 0.3)\%\ \text{U-235} \qquad \text{(Sprague, 1974)}$$

where,

p'_p = fraction of U-235 in re-enriched secondary-produced fuel

p_p = fraction of U-235 in initial primary-produced fuel

These enrichment factors also affect the feed requirement, f', by the relation;

$$f' = \frac{(p_p' - p_t)}{(p_d - p_t)}$$

where,

p_d = fraction of U-235 in depleted uranium from spent fuel

p_t = fraction of U-235 in uranium tails from re-enrichment

Relevant values of the feed requirement f' and separative work requirement s' for various reactors, assuming an enrichment tails assay, p_t, of 0.25% U-235, are presented in table 5.18.

Using this information and energy input results from previous parts of this chapter, the n.e.r.'s of re-cycled uranium fuel, N_B, for different reactors were deduced. Estimates calculated assuming gas diffusion enrichment for all reactors except the PWR, for which the use of gas centrifuge enrichment is additionally assumed, are given in table 5.19. Only results for re-cycled material from reactors which use enriched uranium fuel are included in this table. Reactors such as the MAGNOX and CANDU designs which use natural uranium produce spent fuel which generally contains hardly more U-235 than enrichment tails (0.25% U-235). This depleted uranium is not usually considered for re-cycling.

The amount of energy required to produce plutonium dioxide fuel from spent burner reactor fuel is expressed by the equation;

N_C = net energy requirement of plutonium obtained by reprocessing spent uranium fuel

$$= E_6 + E_7 + E_8$$

The n.e.r.'s of plutonium created by different types of burner reactor were calculated using previously estimated values of the energy inputs; fuel fabrication, E_6, reprocessing, E_7, and

Table 5.18 : Re-enrichment parameters of re-cycled replacement fuel for typical burner reactor designs.

Reactor	Initial fuel content p_p (% U235)	Spent fuel content p_d (% U235)	Re-cycled fuel content p'_p (% U235)	Re-enrichment feed : f' (te/te)	Separative work : s' (te SWU/te U)
ACR :	2.25	0.80	2.55	4.18	2.80
SGHWR :	2.10	0.60	2.40	6.14	2.70
BWR :	2.60	0.75	2.90	5.30	3.60
PWR :	3.10	0.84	3.40	5.34	4.50

Table 5.19 : Net energy requirements for re-cycled uranium dioxide fuel for various burner reactor designs.

Reactor	Net energy requirement : N_B^* (10^6 MJ/te U)			
	minimum		maximum	
	electrical	thermal	electrical	thermal
AGR :	24.22 +	1.53	32.41 +	3.63
SGHWR :	23.55 +	1.97	31.62 +	4.63
BWR :	31.12 +	1.91	41.65 +	4.63
PWR :	38.75 +	2.04	51.80 +	5.05
PWR† :	4.55 +	2.22	6.31 +	6.76

* assumes gas diffusion enrichment except where indicated

† assumes gas centrifuge enrichment

waste disposal, E_8. Results are shown in table 5.20.

The net energy requirement of depleted uranium dioxide fuel which could be utilised by breeder reactors is given by the expression;

N_D = net energy requirement of depleted uranium dioxide obtained by reprocessing spent uranium fuel

$= E_7 + E_8$

Estimates of this n.e.r. are illustrated in table 5.21.

Table 5.20 : Net energy requirements for plutonium dioxide fuel recovered from various burner reactor spent fuels.

Reactor	Net energy requirement : N_C (10^6 MJ/te Pu)			
	minimum		maximum	
	electrical	thermal	electrical	thermal
MAGNOX :	0.21	+ 0.20	0.30	+ 0.53
AGR :	0.25	+ 0.34	0.34	+ 0.64
SGHWR :	0.27	+ 0.36	0.37	+ 0.69
CANDU :	0.24	+ 0.23	0.34	+ 0.58
BWR :	0.28	+ 0.37	0.39	+ 0.72
PWR :	0.28	+ 0.38	0.39	+ 0.72

Table 5.21 : Net energy requirement of depleted uranium dioxide recovered from spent uranium fuel.

Value	Net energy requirement : N_D (10^6 MJ/te U)		
	electrical		thermal
Minimum :	0.066	+	0.099
Maximum :	0.145	+	0.359

6 ANALYSIS OF NUCLEAR POWER GENERATION

6.1 Introduction

Burner reactor power systems generate fuel, usually in the form of electricity, primarily from fission reactions between neutrons and uranium-235 (U-235) nuclei in uranium. Such systems consist of a fuel cycle which includes all processes involved in the manufacture of nuclear fuel from naturally-occurring sources of uranium; a nuclear power station which contains one or more reactors, that provide all necessary conditions for sustaining and controlling fission reactions, together with other equipment for the conversion of heat evolved in the reactor core into electricity; and a transmission and distribution network that delivers electricity from the power station to the consumer.

The system includes all industries, services, etc., which contribute, directly or indirectly, to the production of electricity by burner reactor power stations from uranium. Hence, the net energy requirement of electricity, which excludes the energy content of uranium, consists of all the energy consumed by these activities for the generation and distribution of one unit of electricity. This equals the total energy input to the system per final output of fuel delivered to consumers.

For convenience of analysis, the total energy input to this system can be regarded as the sum of two components; the preliminary and concurrent energy inputs. The preliminary input consists of the energy required by the system prior to actual power generation. This includes energy used in power station construction, initial core fuel fabrication and building

of the electricity supply network. These inputs are similar to capital costs or investment in economics. The concurrent input consists of all the energy consumed by the system during power generation, that is, energy inputs to the operation of the power station, manufacture of replacement fuel and maintenance of the electricity supply network. The economic equivalent of these inputs are operating costs. The fuel output of the system equals the electricity actually available to the consumer at the point of use. Consequently, electricity used internally in the power station and electrical losses during transmission and distribution are excluded from this output.

The purpose of this chapter is to demonstrate how the preliminary and concurrent energy inputs, and the delivered fuel output of typical burner reactor power systems may be evaluated from the results of the previous chapter. The effects of various factors such as uranium ore grade, fuel cycle technology, reactor design and fuel management policy are incorporated into these terms. Results are then used to calculate the net energy requirement of electricity produced by burner reactor power systems.

6.2 Preliminary energy input

The preliminary energy input to a burner reactor power system consists of all the energy required before it can produce electricity. The most important and obvious primary requirements are those involved in the construction of the reactor(s), cooling systems, turbo-generator and other power station equipment; the manufacture and assembly of uranium fuel elements from ores and other sources for the first fuel load, or initial core; and the building of grid and mains networks.

The period of time required to construct a nuclear power station from the preparation of the foundations to actual power production varies widely from site to site, but on average this is about five and a half years (Budwani, 1974). The fuel elements which form the initial core are usually loaded into the reactor about nine months before power generation. This fuel is made from ores mined approximately one and a half years earlier (Nuclear Energy Agency, 1975). Construction of the entire electricity supply network is an apparently perpetual process with components being replaced and improved continuously. However, the time needed to expand the system for a new power plant is usually less than a year.

In general most of the preliminary work on the system occurs in a five or six year period prior to electricity production. The total amount of energy consumed during this period can be summarised by the following expression;

e_p = preliminary energy input per unit net power installed

$$\doteq \quad E_K + (L \times N_A) + E_G$$

where,

E_K = e.r. of power station construction per unit net power installed

L = amount of uranium in the first fuel load per unit net power installed

= initial core inventory

N_A = n.e.r. of primary-produced fuel

E_G = e.r. of electricity supply network construction per unit net power installed

The preliminary input, e_p, is measured in terms of the energy required per actual or 'net' amount of electrical power delivered by, or 'sent out' from, the power station to the grid (eg. MJ/1000MW[e]-so). The net power output equals the total or 'gross' rate at which electricity is generated by the plant minus the amount of power that it uses internally to drive equipment such as cooling fans, pumps, loading machinery, control devices, etc. A typically convenient net power rating of 1000MW[e]-so is used here and the e.r. of power station construction, E_K, initial core inventory, L, and e.r. of electricity supply network construction, E_G, are all standardised to this particular unit.

The amount of energy required to build a power station, E_K, is the sum of the energy consumed in the manufacture, transportation and erection of all materials, equipment and plant used and assembled on site. Six different types of burner reactor power station are considered here; the MAGNOX reactor, Advanced Gas-cooled Reactor (AGR), Steam Generating Heavy Water Reactor (SGHWR), CANDU reactor, Boiling Water Reactor (BWR) and Pressurised Water Reactor (PWR). Details of the analysis of their construction are presented in appendix L and the results are summarised in table 6.1.

The energy input of the initial core inventory depends on the amount of uranium in the first loading, L, and the energy required to produce it, N_A. Table 6.2 indicates estimates of the average core inventories for typical reactor designs. These data, which correspond to fuel enrichment parameters given in table 5.16, were obtained from standard reactor directories and indexes (International Atomic Energy Agency, 1967, 1970a, 1972; Nuclear Engineering International, 1972).

Table 6.1 : Estimated energy requirements for power station construction.

Reactor	Energy requirement : E_K (10^6 MJ/1000MW(e)-so)	
	minimum	maximum
MAGNOX :	855(e) + 9895(t)	1435(e) + 13420(t)
AGR :	595(e) + 7095(t)	870(e) + 9650(t)
SGHWR :	1135(e) + 13270(t)	1630(e) + 20235(t)
CANDU :	2110(e) + 27175(t)	3685(e) + 42250(t)
BWR :	590(e) + 4715(t)	1080(e) + 8150(t)
PWR :	590(e) + 4715(t)	1080(e) + 8150(t)

Table 6.2 : Average initial core inventories for typical reactors.*

Reactor	Initial core inventory : L (te U/1000MW(e)-so)
MAGNOX :	1010
AGR :	195
SGHWR :	160
CANDU :	167
BWR :	141
PWR :	101

* values correspond to the specific enrichment parameters given in table 5.16.

The nuclear fuel n.e.r. used to evaluate the preliminary energy contribution of the core is that of primary-produced fuel, N_A, since it is not common practice to incorporate re-cycled fuel in initial loadings. Values of the n.e.r. of primary-produced fuel for different types of reactor are shown in table 5.17.

Estimates of the energy required to construct transmission and distribution networks between the power station and consumer, E_G, are deduced in appendix M for a typical electricity supply system. Results are presented in table 6.3. Using all this information, values of the total preliminary energy input to burner reactor power systems were calculated and subsequent results are given in table 6.4.

6.3 Concurrent energy input

The concurrent energy input to a burner reactor power system consists of all energy used during the production of electricity from the system. The input includes the energy required to manufacture replacement fuel, and to maintain and repair the reactor, other parts of the power station and the electricity supply network. Excluded from the concurrent input is the electricity used internally by the power station and power lost during transmission and distribution. This electricity is directly deducted from the gross electrical output of the system (see 6.4).

There are three specific contributions to the concurrent energy input; replacement of moderator material losses, system maintenance and repair, and refuelling. Moderator losses occur because of leakage, degradation, etc., and these must be restored to ensure that the reactor continues to operate in the designated manner. System maintenance and repair involves

Table 6.3 : Estimated energy requirements for electricity supply network construction.

Value	Energy requirement : E_G (10^6 MJ/1000MW[e]-so)		
Minimum :	125(e)	+	1700(t)
Maximum :	535(e)	+	5530(t)

Table 6.4 : Preliminary energy inputs for various burner reactor power systems.

Reactor	Preliminary energy input : e_p^* (10^6 MJ/1000MW[e]-so)			
	minimum		maximum	
	electrical	thermal	electrical	thermal
MAGNOX	$(\frac{6.36}{G} + 1190) + (\frac{20.6}{G} + 11800)$		$(\frac{856}{G} + 2190) + (\frac{1120}{G} + 23100)$	
AGR	$(\frac{3.72}{G} + 3010) + (\frac{12.1}{G} + 8970)$		$(\frac{505}{G} + 4440) + (\frac{659}{G} + 17700)$	
SGHWR	$(\frac{4.00}{G} + 4170) + (\frac{13.1}{G} + 15200)$		$(\frac{542}{G} + 6020) + (\frac{710}{G} + 28500)$	
CANDU	$(\frac{1.05}{G} + 2270) + (\frac{3.41}{G} + 28900)$		$(\frac{142}{G} + 4260) + (\frac{185}{G} + 48400)$	
BWR	$(\frac{3.78}{G} + 3520) + (\frac{12.4}{G} + 6590)$		$(\frac{515}{G} + 5330) + (\frac{672}{G} + 16300)$	
PWR	$(\frac{3.06}{G} + 3150) + (\frac{10.0}{G} + 6550)$		$(\frac{415}{G} + 4850) + (\frac{543}{G} + 15800)$	
PWR†	$(\frac{3.06}{G} + 947) + (\frac{10.0}{G} + 6570)$		$(\frac{415}{G} + 1990) + (\frac{543}{G} + 15900)$	

* assumes gas diffusion enrichment of fuel except where indicated otherwise.

† assumes gas centrifuge enrichment of fuel.

Note, G = ore grade in % U_3O_8.

preventing and correcting faults by checking, adjusting and replacing equipment and machinery.

The removal of spent fuel from the reactor core and its replacement with new fuel is known as refuelling or reloading and this is a very important concurrent input. Although reactor type determines the frequency of refuelling, general fuel management policies decide the particular composition of the new fuel. This can consist of primary-produced fuel obtained entirely from naturally-occurring sources of uranium or secondary-produced fuel that contains re-cycled uranium and/or plutonium recovered from previously irradiated, spent fuel. Since the original source and composition of replacement fuel affects the concurrent energy input to the system, different basic fuel management policies are treated separately here. Policies which involve primary-produced fuel only are examined under the heading of the 'primary fuel cycle', whilst those which use secondary-produced fuel are investigated in the 'secondary fuel cycle'.

6.3.1 Primary fuel cycle

The primary fuel cycle is the most widespread fuel management policy in use at the moment. This consists of refuelling reactors with uranium produced from resources by a sequence of operations involving exploration, ore mining and processing, refining, conversion, enrichment (when required) and fuel fabrication. The total concurrent energy input to the reactor power system throughout its entire operational life can be represented by the following equation;

e_c = concurrent energy input per unit net power installed for the primary fuel cycle

$$= \left[(M \times E_M) + (A \times N_A) + E_N \right] \times y \times l_f$$

where,

M = annual moderator replacement rate per unit net power installed

E_M = e.r. of moderator material

A = amount of uranium incorporated into replacement fuel each year per unit net power installed

= annual refuelling rate

N_A = n.e.r. of primary-produced uranium fuel

E_N = e.r. of electricity supply system annual maintenance and repair per unit net power installed

y = power station operating life in years

l_f = fraction of the operating life that the system produces electricity

= load factor

As indicated the total concurrent energy input consists of three important contributions; the energy inputs of replacing moderator losses, refuelling, and maintenance and repair.

The energy required to compensate moderator losses depends on the replacement rate, M, and the moderator e.r., E_M. Although all reactor designs are subject to some form of moderator loss, replacement is only feasible for certain types of reactor. The solid graphite moderator material of MAGNOX and AGR systems can suffer losses of up to almost 0.5% of the total mass each year (Central Electricity Generating Board, 1971), but it is usually impractical to compensate such losses. BWR and PWR designs can also leak cooling and moderating water but

the replacement of such losses is generally regarded as unimportant, in economic and energy terms, since 'light' water coolant and moderator is relatively inexpensive and uses little fuel to produce. SGHWR and CANDU reactors, however, rely on expensive, fuel consuming, 'heavy' water or, deuterium oxide, and hence compensating the yearly loss of between 0.2% and 0.4% of the total volume (Moore, Hicks, Bradley and Rowlands, 1973; Canadian Nuclear Agency, 1974) can be very significant. Average moderator losses, replacement rates and estimated values of e.r.'s for moderator materials are summarised in table 6.5.

The energy contribution of replacing spent fuel with material from the primary fuel cycle equals the product of the annual refuelling rate, A, and the n.e.r. of primary-produced fuel, N_A. Both these terms are influenced by reactor design. Average values of the refuelling rate for typical reactor systems with fuel enrichment parameters given in table 5.16 were deduced from operating data (International Atomic Energy Agency, 1967, 1970a, 1972; Nuclear Engineering International, 1972) and results are shown in table 6.6. Estimates of fuel n.e.r.'s evaluated in the previous chapter are illustrated in table 5.17. The energy requirement of keeping the electricity supply network in good working order, E_N, was assessed in appendix L and results are summarised in table 6.7.

The sum of all three energy contributions equals the annual concurrent energy input. The total energy required during the entire period that the reactor power system generates and delivers electricity can be found by multiplying this annual input by the estimated operational life of the power station, y, and the load factor, l_f. Estimates of the average plant

Table 6.5 : Average moderator replacement requirements for typical reactor designs.

Reactor	Loss rate (% total mass per year)	Replacement rate : M (kg/1000MW[e]-so/year)	Energy requirement : E_M (MJ/kg)	
			electrical	thermal
MAGNOX :	0.15	0	30±14	+ 86±27
AGR :	0.15	0	30±14	+ 86±27
SGHWR :	0.40	920	2200±600	+ 28000±6200
CANDU :	0.20	1020	2200±600	+ 28000±6200
BWR :	n.a.	n.a.	0.002	
PWR :	n.a.	n.a.	0.002	

n.a. = not available

Table 6.6 : Average annual refuelling rates for typical reactors.*

Reactor	Annual refuelling rate : A (te U/1000MW[e]-so/year)
MAGNOX :	322
AGR :	48.5
SGHWR :	53.5
CANDU :	127
BWR :	35.3
PWR :	33.8

* corresponding to fuel enrichment parameters given in table 5.16.

Table 6.7 : Estimated energy requirements for the maintenance and repair of the electricity supply network.

Value	Energy requirement : E_N (10^6 MJ/1000MW(e)-so/year)
Minimum :	120(t)
Maximum :	5(e) + 140(t)

lifetime obtained from reactor manuals (International Atomic Energy Agency, 1967, 1970a, 1972; Nuclear Engineering International, 1972; Nuclear Energy Agency, 1975) are presented in table 6.8. The load factor, or fraction of this lifetime that the system actually produces electricity, is determined by operational considerations such as the time required for routine maintenance, unexpected repairs, etc. This parameter is an independent variable and consequently the total concurrent energy input is measured as the energy required per unit load factor, e_c/l_f. Values of this result are given in table 6.9.

6.3.2 Secondary fuel cycle

Secondary fuel cycle policies consist of refuelling reactors with fissile-bearing material such as uranium and plutonium that has been recovered from previously-irradiated, or spent, fuel by reprocessing. There are many different types of secondary fuel cycle policies but only two basic varieties are considered here; 'single-stage' uranium re-cycling and 'uranium/plutonium mixed oxide' re-cycling.

Single-stage uranium re-cycling involves refuelling with fuel containing uranium that has previously been used only once in the reactor. Such a fuel policy avoids probable complications caused by the accumulation, from repeated, or 'multi-stage', re-cycling, of artificially created isotopes such as neutron absorbing uranium-236 (U-236) in the fuel. Single-stage re-cycling would be achieved by careful fuel management which ensures that recovered uranium would be removed from the fuel cycle immediately after it had been re-irradiated in the reactor.

Table 6.8 : Average reactor power station operational lifetimes.

Reactor	Operating life : y (years)
MAGNOX :	20.0
AGR :	22.5
SGHWR :	30.0
CANDU :	30.0
BWR :	35.0
PWR :	35.0

Table 6.9 : Concurrent energy inputs per unit load factor for the primary fuel cycle of various burner reactor power systems.

Reactor	Concurrent energy input per unit load factor : e_c/l_f* (10^6 MJ/1000MW[e]-so)			
	minimum		maximum	
	electrical	thermal	electrical	thermal
MAGNOX	$\frac{(40.6+1330)}{G} + \frac{(131+3490)}{G}$		$\frac{(5460+1480)}{G} + \frac{(7150+29200)}{G}$	
AGR	$\frac{(29.7+21700)}{G} + \frac{(96.9+4010)}{G}$		$\frac{(4030+28800)}{G} + \frac{(5270+23200)}{G}$	
SGHWR	$\frac{(40.1+29100)}{G} + \frac{(131+6030)}{G}$		$\frac{(5440+39000)}{G} + \frac{(7130+32400)}{G}$	
CANDU	$\frac{(24.0+777)}{G} + \frac{(77.7+4810)}{G}$		$\frac{(3230+1180)}{G} + \frac{(4230+20300)}{G}$	
BWR	$\frac{(39.0+31900)}{G} + \frac{(127+5980)}{G}$		$\frac{(5290+42600)}{G} + \frac{(6920+31500)}{G}$	
PWR	$\frac{(45.5+40600)}{G} + \frac{(148+6530)}{G}$		$\frac{(6180+54200)}{G} + \frac{(8070+36000)}{G}$	
PWR†	$\frac{(45.5+3740)}{G} + \frac{(148+6450)}{G}$		$\frac{(6180+6330)}{G} + \frac{(8070+37800)}{G}$	

* assumes gas diffusion enrichment of fuel except where indicated otherwise.

† assumes gas centrifuge enrichment of fuel.

Note, G = ore grade in % U_3O_8.

The total concurrent energy input throughout the operating life of the reactor due to single-stage uranium re-cycle refuelling, moderator loss replacement, and system maintenance and repair can be summarised by the equation;

e'_c = concurrent energy input per unit net power installed for the secondary fuel cycle with single-stage uranium re-cycling.

$$= \left[\left(M \times E_M \right) + \left(A - \frac{B}{f'} \right) \times N_A + \left(\frac{B}{f'} \times N_B \right) + E_N \right] \times y \times l_f$$

where,

M = annual moderator replacement rate per unit net power installed

E_M = e,r. of moderator material

A = amount of primary-produced uranium incorporated into replacement fuel each year per unit net power installed for a primary fuel cycle (see 6.3.1)

= standard annual refuelling rate

B = amount of depleted uranium in irradiated fuel removed each year per unit net power installed

= annual spent fuel output rate

f' = amount of depleted, irradiated uranium feed per unit mass re-enriched re-cycled uranium

N_A = n.e.r. of primary-produced uranium fuel

N_B = n.e.r. of secondary-produced uranium fuel

E_N = e.r. of electricity supply system annual maintenance and repair per unit net power installed

y = power station operating life in years

l_f = load factor

The energy contribution of moderator replacement, $M \times E_M$, and system maintenance and repair, E_N, are the same as for the primary fuel cycle (see 6.3.1). These inputs can be deduced from results given previously in table 6.5 and 6.7.

The energy requirement of refuelling is determined by the relative amount of each type of fuel used in replacement loads. The standard refuelling rate, A, is the amount of uranium required in the fuel if a primary fuel cycle were being used (see 6.3.1). With a secondary fuel cycle the amount of primary-produced fuel equals the standard refuelling rate, A, less the amount of secondary-produced fuel introduced by re-cycling, B/f' (see figure 6.1). The quantity of secondary-produced fuel available for re-cycling is determined by the amount of depleted uranium recovered from spent fuel, B, and the amount required to obtain a unit mass of re-enriched uranium, f' (see table 5.18). The values of these parameters depend on reactor design and typical results deduced from reactor operating data (International Atomic Energy Agency, 1967, 1970a, 1972; Nuclear Engineering International, 1972; Nuclear Energy Agency, 1975) are illustrated in table 6.10. Only reactor systems which use enriched fuel are included since highly-depleted fuel from natural uranium reactors (MAGNOX and CANDU) is not generally considered suitable for re-cycling (see 5.5.2).

The total concurrent energy input per unit load factor for single-stage uranium re-cycling, e'_c/l_f, was evaluated by combining this information with estimates of the n.e.r.'s for primary- and secondary-produced uranium fuel, N_A (see table 5.17) and N_B (see table 5.19) respectively, and values of the power station lifetimes, y (see table 6.8). Subsequent results for e'_c/l_f are presented in table 6.11.

Figure 6.1 : Schematic diagram of the secondary fuel cycle with single-stage uranium re-cycling.

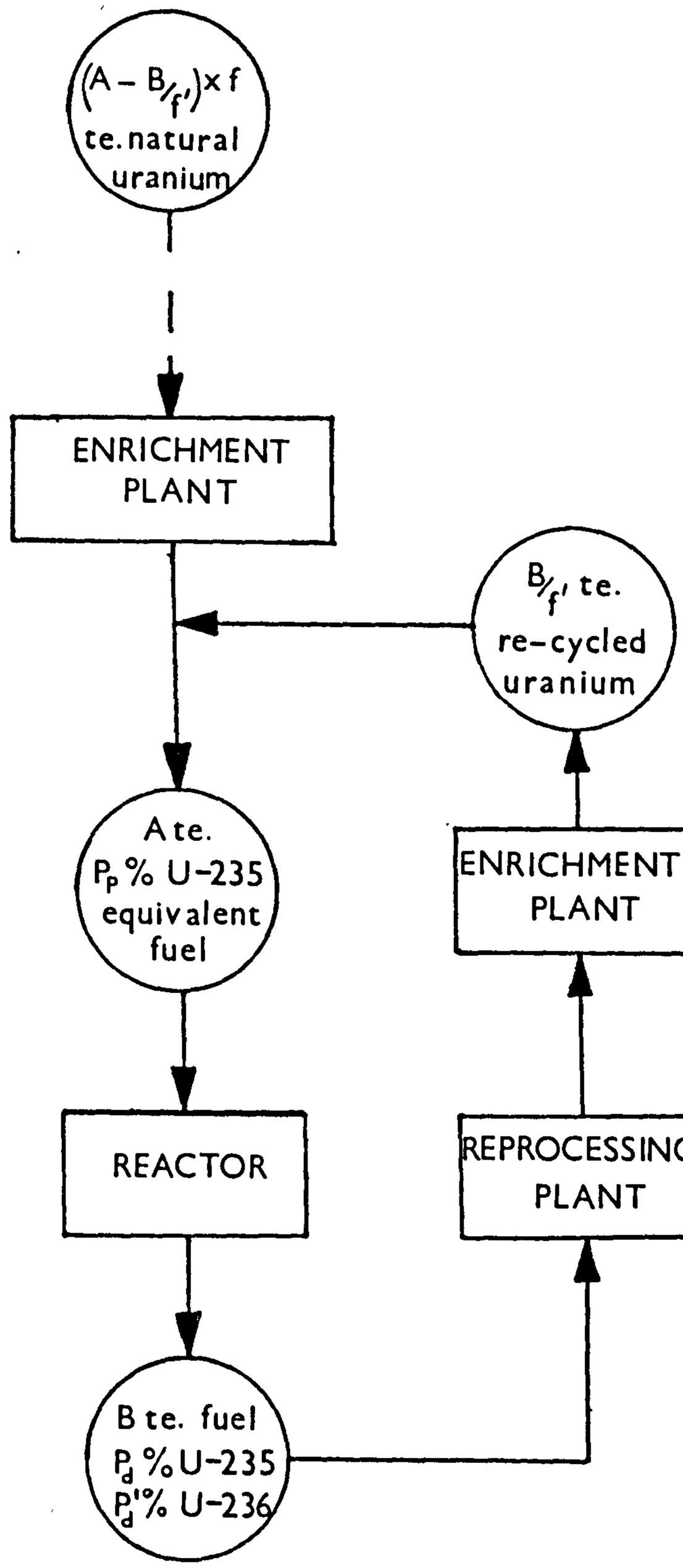


Table 6.10 : Average annual refuelling parameters for typical reactors using single-stage uranium fuel re-cycling.*

Reactor	Standard annual refuelling rate : A (te U/1000MW[e]-so/year)	Annual spent uranium output rate : B (te U/1000MW[e]-so/year)	Re-enrichment feed requirement rate : f' (te U/te U)	Secondary produced fuel input : $\frac{B}{f'}$ (te U/1000MW[e]-so/year)	Primary produced fuel input : $\frac{(A-B)}{f'}$ (te U/1000MW[e]-so/year)
AGR :	48.5	47.4	4.18	11.3	37.2
SGHWR :	53.5	52.1	6.14	8.5	45.0
BWR :	35.3	34.0	5.30	6.4	28.9
PWR :	33.8	32.5	5.34	6.1	27.7

* corresponding to fuel enrichment parameters given in table 5.18.

Table 6.11 : Concurrent energy inputs per unit load factor for the secondary fuel cycle of various burner reactor power systems incorporating single-stage uranium re-cycling.

Reactor	Concurrent energy input per unit load factor : e'_c/l_f* (10^6 MJ/1000MW[e]-so)			
	minimum		maximum	
	electrical	thermal	electrical	thermal
AGR	$\frac{(22.8+22800)}{G} + \frac{(74.3+4090)}{G}$		$\frac{(3090+30400)}{G} + \frac{(4040+19500)}{G}$	
SGHWR	$\frac{(33.7+30500)}{G} + \frac{(110+6240)}{G}$		$\frac{(4580+40900)}{G} + \frac{(5990+15700)}{G}$	
BWR	$\frac{(32.0+33100)}{G} + \frac{(104+6080)}{G}$		$\frac{(4330+44200)}{G} + \frac{(5660+27700)}{G}$	
PWR	$\frac{(37.3+41600)}{G} + \frac{(122+6320)}{G}$		$\frac{(5060+55500)}{G} + \frac{(6610+31500)}{G}$	
PWR†	$\frac{(37.3+4030)}{G} + \frac{(122+6520)}{G}$		$\frac{(5060+6560)}{G} + \frac{(6610+33300)}{G}$	

* assumes gas diffusion enrichment of fuel except where indicated otherwise

† assumes gas centrifuge enrichment

Note, G = ore grade in % U_3O_8.

Mixed oxide fuel contains secondary-produced re-cycled uranium and plutonium as well as primary-produced uranium. Although plutonium is usually regarded as a basic fuel for breeder reactors, some interest has been shown in the possibility of using such fissile material in burner reactors, in particular Light Water Reactor (LWR) designs of the USA which include BWR and PWR systems (eg. Hnilica, Holley, Lahner and Schmale, 1974; Puechl, 1975). Assuming a single-stage re-cycling policy in which reprocessed uranium and plutonium was only re-irradiated once, then the total concurrent energy input to a system using this type of refuelling can be expressed by the following relationship;

e''_C = concurrent energy input per unit net power installed for the secondary fuel cycle with single-stage uranium and plutonium mixed oxide re-cycling

$$= \left[(M \times E_M) + (A - \frac{B}{f'} - C') \times N_A + (\frac{B}{f'} \times N_B) + (C \times N_C) + E_N\right] \times y \times l_f$$

where,

M = annual moderator replacement rate per unit net power installed

E_M = e.r. of moderator material

A = standard annual uranium refuelling rate (see 6.3.1)

B = annual spent uranium fuel output rate

C = amount of plutonium in irradiated fuel removed from the reactor each year per unit net power installed

= annual plutonium production rate

C' = equivalent amount of uranium fuel replaced by plutonium re-cycling each year per unit net power installed

= annual uranium fuel equivalent plutonium re-cycling rate

f' = amount of depleted uranium feed per unit mass re-enriched, re-cycled uranium

N_A = n.e.r. of primary-produced uranium fuel

N_B = n.e.r. of secondary-produced uranium fuel

N_C = n.e.r. of re-cycled plutonium

E_N = e.r. of electricity supply system annual maintenance and repair per unit net power installed

y = power station operating life in years

l_f = load factor

The contribution of moderator loss replacement, M x E_M, and electricity network maintenance, E_N, to the concurrent energy input, e''_c, are the same as for the primary fuel cycle (see table 6.5 and table 6.7 respectively). The lifetime factors, y, are also the same as those given previously in table 6.8.

The energy contribution of mixed oxide refuelling depends on the amount of uranium fuel that would normally be required by the reactor with the primary fuel cycle, that is A, and the amount of depleted uranium, B, and plutonium, C, obtained from spent fuel. The quantity of primary-produced uranium in the fuel equals the standard amount, A, less the amount of re-enriched, re-cycled uranium, B/f', and the amount of uranium substituted by re-cycled plutonium, C'. Values of these refuelling parameters for the PWR design using the type of fuel described by table 5.18 were deduced from reactor data (International Atomic Energy Agency, 1972; Nuclear Energy Agency, 1975; Puechl, 1975). These parameters are illustrated in figure 6.2 and summarised in table 6.12.

Using estimated n.e.r.'s, N_A, N_B and N_C evaluated earlier (see tables 5.17, 5.19 and 5.20) and reactor operating information,

Figure 6.2 : Schematic diagram of the secondary fuel cycle with single-stage mixed oxide uranium and plutonium re-cycling for a typical PWR design.

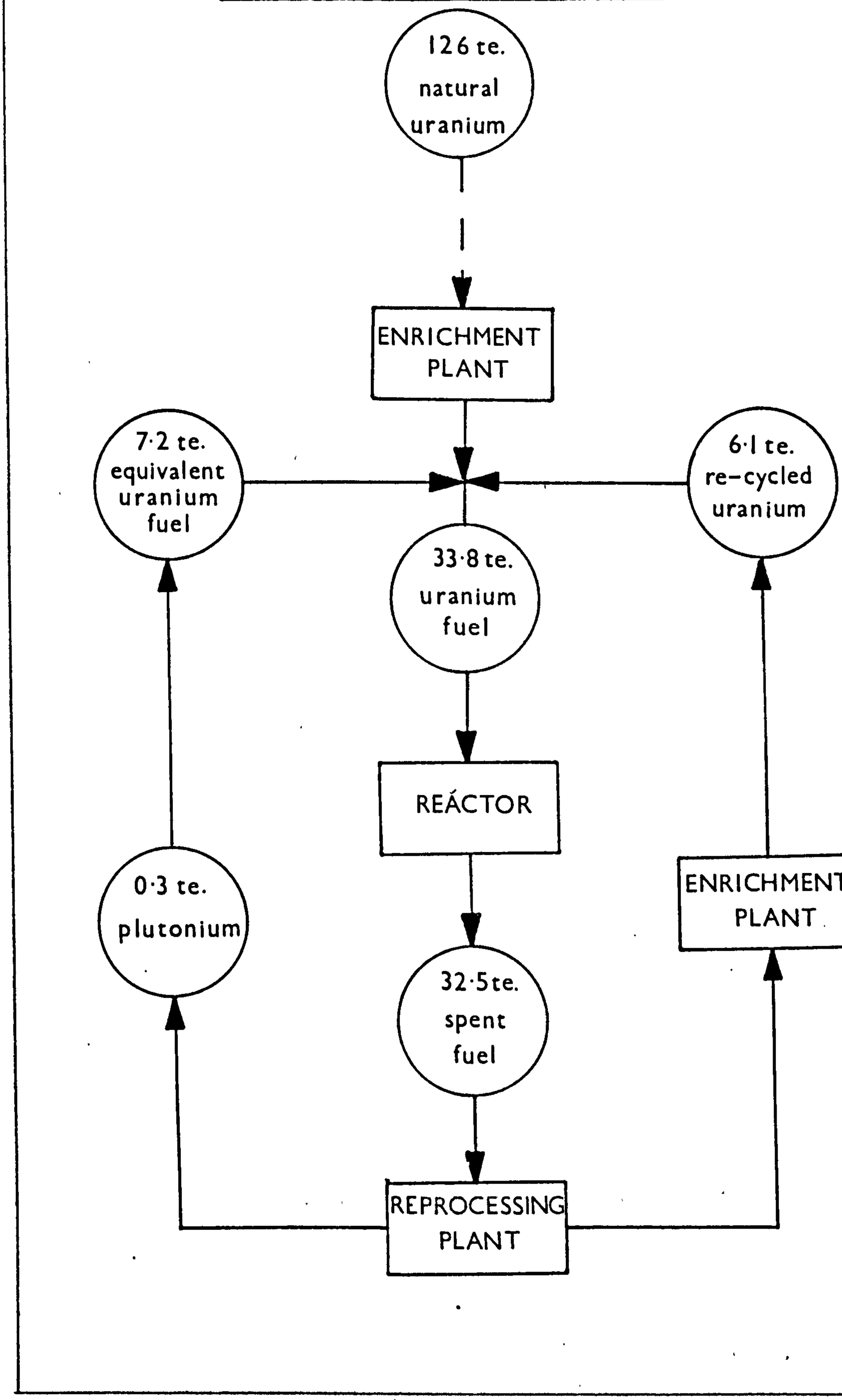


Table 6.12 : Average annual refuelling parameters for a PWR system using single-stage mixed oxide uranium and plutonium re-cycling.*

Parameter		Units		Value
Standard annual uranium refuelling rate : A	:	te U/1000MW(e)-so/year	:	33.8
Annual spent uranium output rate : B	:	te U/1000MW(e)-so/year	:	32.5
Annual plutonium production rate : C	:	te Pu/1000MW(e)-so/year	:	0.30
Annual uranium fuel equivalent re-cycling rate : C'	:	te U/1000MW(e)-so/year	:	7.20
Re-enrichment feed requirement : f'	:	te depleted U/te re-enriched U	:	5.34
Secondary-produced uranium input : $\frac{B}{f'}$	:	te U/1000MW(e)-so/year	:	6.10
Primary-produced uranium input : $(A - \frac{B}{f'} - C')$	:	te U/1000MW(e)-so/year	:	20.5

* corresponding to fuel enrichment parameters given in table 5.18.

the energy requirement of refuelling was calculated. From this the total concurrent energy input per unit load factor, e''_c/l_f, was deduced and results for the PWR design are presented in table 6.13.

6.4 Delivered fuel output

The total amount of fuel actually delivered to consumers by any fuel supply system depends on many factors. In a burner reactor power system neutron losses, spurious reactions, etc., cause differences between the amount of heat, or gross thermal output, available from the nuclear fuel in the reactor core and the theoretical energy content of uranium. Additionally, energy transfer inefficiencies and heat losses reduce the amount of heat produced by the reactor cooling system, or nuclear steam supply system (NSSS).

Although 'nuclear steam' obtained from the reactor could be used for industrial process heating (Canadian Nuclear Agency, 1973; Nuclear Engineering International, 1976), it is generally converted to electricity by turbo-generator equipment. Due to inefficiencies in conversion the amount of electrical energy, or gross electrical output, generated is less than the initial heat energy input. Energy losses occur as waste heat which could be used as a low-grade source of energy, although this is not a common practice.

The amount of electricity delivered by the nuclear reactor power station to the transmission network, or grid, is fractionally less than the gross electrical output, since some electricity is used internally to drive power plant machinery. This reduced output is usually referred to as the, net or 'sent out', electrical output. Energy losses occur during

Table 6.13 : Concurrent energy input per unit load factor for the secondary fuel cycle of a PWR system incorporating single-stage mixed oxide uranium and plutonium re-cycling.

Enrichment method Concurrent energy input per unit load factor : e''_c/l_f

(10^6 MJ/1000MW[e] -so)

	minimum		maximum	
	electrical	thermal	electrical	thermal
Gas diffusion :	$(\frac{27.6}{G}+32900)$	$+ (\frac{90.0}{G}+5890)$	$(\frac{3740}{G}+44000)$	$+ (\frac{4890}{G}+24100)$
Gas centrifuge:	$(\frac{27.6}{G}+3240)$	$+ (\frac{90.0}{G}+6040)$	$(\frac{3740}{G}+5260)$	$+ (\frac{4890}{G}+26300)$

transmission and distribution, and, in addition, electricity is consumed by machinery and plant in the supply network itself. Consequently the amount of electricity made available to consumers, that is, the delivered fuel output, is less than the net electrical output of the power station.

The total amount of electricity, in mega-joules, produced and delivered by a given size of power station throughout its operational lifetime can be represented by the following expression;

e_o = delivered fuel output per unit net installed power

$= y \times l_f \times e_e \times 3.1536 \times 10^{10}$ MJ(e)/1000MW[e]-so

where,

y = power station operating life in years

l_f = load factor

e_e = ratio of delivered fuel output to net electrical output

= electricity supply network efficiency

The delivered fuel output, e_o, is written in terms of the energy output per unit power, or rate of electricity production, of the power station. The power rating is standardised to a net value of 1000 MW (10^9 watts) of electricity and consequently this way of measuring the output automatically accounts for all inefficiencies, losses and fuel use that occur in the power station itself.

The total output, e_o, is obtained by multiplying the lifetime, y, by the load factor, l_f, and the electricity supply network efficiency, e_e. The lifetime factor indicates the number of years that the power station exists as a viable piece of industrial equipment and values for various reactor designs are

given in table 6.8. The fraction of the net power available to consumers, or network efficiency, e_e, depends on the nature of the transmission and distribution system in use. Analysis of a typical electricity supply network is presented in appendix M and the efficiency is estimated as approximately 0.90. The power rating of the power station in mega-watts is converted to an annual energy output in mega-joules by a conversion factor of 3.1536×10^7 MJ per MW-year.

Since the fraction of time that the power station can produce electricity, or load factor, l_f, is an independent variable, values of the delivered fuel output were deduced in terms of energy per unit load factor, e_o/l_f. Results for the different types of reactor studied here are illustrated in table 6.14.

6.5 Net energy requirement of electricity

The net energy requirement of electricity equals all energy consumed in the generation and delivery of that fuel excluding the energy content of the particular energy resource from which it is originally produced. The net energy requirement of electricity obtained from burner reactor power systems can be deduced from the simplified terms describing the energy inputs and fuel output of these systems;

$$R = \text{net energy requirement of electricity}$$

$$= \frac{(e_p + e_c)}{e_o} \qquad \text{MJ(t)/MJ(e)}$$

where,

e_p = preliminary energy input per unit net power installed

e_c = concurrent energy input per unit net power installed

e_o = delivered fuel output per unit net power installed

Table 6.14 : Average values of the lifetime delivered fuel output for various burner reactor power systems.

Reactor	Delivered fuel output per unit load factor : e_o/l_f (10^{11} MJ/1000MW(e) -so)
MAGNOX :	5.68
AGR :	6.39
SGHWR :	8.51
CANDU :	8.51
BWR :	9.93
PWR :	9.93

To evaluate the n.e.r. of electricity, R, using estimates of these inputs and outputs calculated in previous sections, it is necessary to establish the specific nature of the system under investigation. Values of the energy inputs e_p and e_c presented in tables 6.4, 6.9, 6.11 and 6.13 are written in terms of electrical energy, (e), and the energy from fossil fuels, or thermal energy, (t). If the n.e.r., R, is required to indicate the total amount of fossil fuel used by the burner reactor power system to produce and deliver one unit of electricity, then the electrical energy input must be converted to a thermal energy input. The factor for converting electrical energy to thermal energy depends on the original source of the electricity. Two particular sources will be considered here - a power system which uses fossil fuels such as coal, oil and gas, and the burner reactor power system itself.

A situation in which most or all the electricity available is obtained from fossil fuel-fired power plant is common in many industrialised countries. The total amount of energy from fossil fuel resources used to produce one unit of electricity, or the gross energy requirement of fossil fuel-generated electricity, is approximately 4 MJ(t) per MJ(e) (see table 3.1 and Chapman, 1973b). Using this conversion factor the energy inputs to the burner reactor power system can be expressed in the same units of MJ(t) and the net energy requirement of nuclear-generated electricity, R, can be evaluated. Such results for different reactor designs using the primary fuel cycle are shown in table 6.15.

The variation of this n.e.r., R, with ore grade, G, for a PWR system is illustrated in figure 6.3. The PWR system was chosen for this analysis because such designs currently comprise the

Table 6.15 : Net energy requirements of electricity from burner reactor power systems using the primary fuel cycle and electrical energy obtained from fossil fuel-fired power plant.*

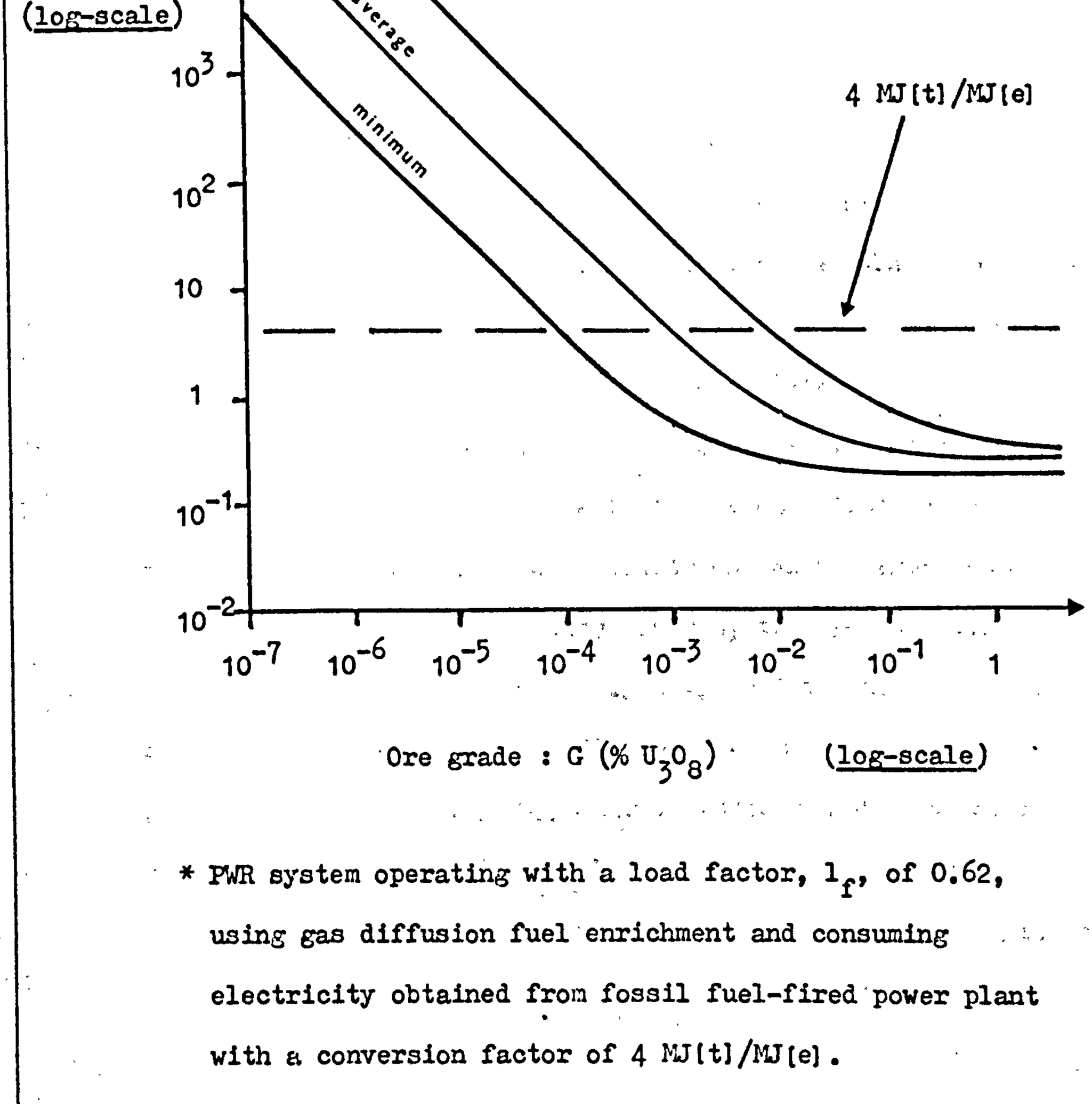
Reactor	Value	Net energy requirement‡ : R (MJ[t]/MJ[e])
MAGNOX	minimum	$\frac{8.1\times10^{-5}}{G} + 0.029)/_{1_f} + (\frac{5.2\times10^{-4}}{G} + 0.015)$
	maximum	$\frac{8.0\times10^{-3}}{G} + 0.056)/_{1_f} + (\frac{0.051}{G} + 0.062)$
AGR	minimum	$\frac{4.2\times10^{-5}}{G} + 0.033)/_{1_f} + (\frac{3.4\times10^{-4}}{G} + 0.142)$
	maximum	$\frac{4.2\times10^{-3}}{G} + 0.055)/_{1_f} + (\frac{0.033}{G} + 0.216)$
SGHWR	minimum	$\frac{3.4\times10^{-5}}{G} + 0.037)/_{1_f} + (\frac{3.4\times10^{-4}}{G} + 0.144)$
	maximum	$\frac{3.3\times10^{-3}}{G} + 0.062)/_{1_f} + (\frac{0.034}{G} + 0.221)$
CANDU	minimum	$\frac{8.9\times10^{-6}}{G} + 0.045)/_{1_f} + (\frac{2.0\times10^{-4}}{G} + 0.009)$
	maximum	$\frac{8.8\times10^{-4}}{G} + 0.077)/_{1_f} + (\frac{0.020}{G} + 0.029)$
BWR	minimum	$\frac{2.8\times10^{-5}}{G} + 0.021)/_{1_f} + (\frac{2.8\times10^{-4}}{G} + 0.135)$
	maximum	$\frac{2.7\times10^{-3}}{G} + 0.038)/_{1_f} + (\frac{0.028}{G} + 0.203)$
PWR	minimum	$\frac{2.2\times10^{-5}}{G} + 0.019)/_{1_f} + (\frac{3.3\times10^{-4}}{G} + 0.170)$
	maximum	$\frac{2.2\times10^{-3}}{G} + 0.035)/_{1_f} + (\frac{0.033}{G} + 0.255)$
PWR†	minimum	$\frac{2.2\times10^{-5}}{G} + 0.010)/_{1_f} + (\frac{3.3\times10^{-4}}{G} + 0.022)$
	maximum	$\frac{2.2\times10^{-3}}{G} + 0.024)/_{1_f} + (\frac{0.033}{G} + 0.064)$

* assumes all electricity used within the system is supplied by fossil fuel-fired power plant, i.e. electrical conversion factor = 4 MJ(t) per MJ(e).

‡ assumes enrichment by gas diffusion except where indicated otherwise.

† assumes enrichment by gas centrifuge.

Figure 6.3 : Ore grade variation of the net energy requirement of electricity produced by a typical burner reactor power system using the primary fuel cycle and electrical energy from fossil fuel-fired power plant.*



* PWR system operating with a load factor, l_f, of 0.62, using gas diffusion fuel enrichment and consuming electricity obtained from fossil fuel-fired power plant with a conversion factor of 4 MJ[t]/MJ[e].

majority of all types of reactor in operation throughout the world at the moment (Nuclear Engineering International, 1972). The use of the widespread gas diffusion technique for fuel enrichment and a typical operational load factor, l_f, of 0.62 (see 7.2 and Searby, 1971) were also assumed in calculating the data for this diagram. In addition to the minimum and maximum variation, the 'average' variation of R with ore grade is shown in figure 6.3. This average result incorporates the logarithmic mean energy input from ore mining and processing operations and the arithmetic mean energy inputs from all other activities. Evidence supporting the use of a logarithmic mean result to represent typical data for uranium concentrate production is outlined in section 5.2.4.

Equations for the average net energy requirement of electricity produced by all the main burner reactor designs are given in table 6.16. From such expressions and the information illustrated in figure 6.3 it is possible to calculate the point at which the amount of fossil fuel required to produce one unit of electricity from burner reactor power systems is the same as that required by a fossil fuel-fired power system, that is, where R equals 4 MJ(t) per MJ(e). With the primary fuel cycle, gas diffusion enrichment and using electrical energy from 'external' (fossil fuel-fired power plant) sources, this point occurs when the grade of ore consumed by a typical reactor system is between 1 and 100 parts per million triuranium octoxide (10^{-6}: ppm U_3O_8), on average , 10ppm U_3O_8. Consequently with ores richer than about 10 ppm U_3O_8 burner reactor power systems consume less fossil fuel than coal-, oil- and gas-burning power systems, whilst for leaner ores the situation is reversed.

Table 6.16 : Average net energy requirements of electricity from burner reactor power systems using the primary fuel cycle, a load factor of 0.62 and electrical energy obtained from fossil fuel-fired power plant.*

Reactor	Net energy requirement‡ : R (MJ[t]/MJ[e])
MAGNOX :	$\frac{6.4\times10^{-3}}{G} + 0.107$
AGR :	$\frac{4.0\times10^{-3}}{G} + 0.250$
SGHWR :	$\frac{3.9\times10^{-3}}{G} + 0.262$
CANDU :	$\frac{2.1\times10^{-3}}{G} + 0.117$
BWR :	$\frac{3.2\times10^{-3}}{G} + 0.217$
PWR :	$\frac{3.6\times10^{-3}}{G} + 0.256$
PWR† :	$\frac{3.6\times10^{-3}}{G} + 0.070$

* assumes all electricity used within the system is supplied by fossil fuel-fired power plant, i.e. electrical conversion factor = 4MJ(t) per MJ(e).

‡ assumes enrichment by the gas diffusion technique except where indicated otherwise.

† assumes enrichment by the gas centrifuge technique.

Note, G = ore grade in % U_3O_8.

Although many national fuel supply systems use a number of different energy resources to produce electricity, it would be possible to re-structure these systems so that all electricity was obtained from a single source. In such a case fuel supply systems would be likely to consume a certain part of their own output for fuel. To evaluate the net energy requirement of electricity from a burner reactor power system that uses such 'self-generated' electrical energy, it is necessary to modify the way in which this result is calculated from the energy inputs e_p and e_c and the fuel output e_o.

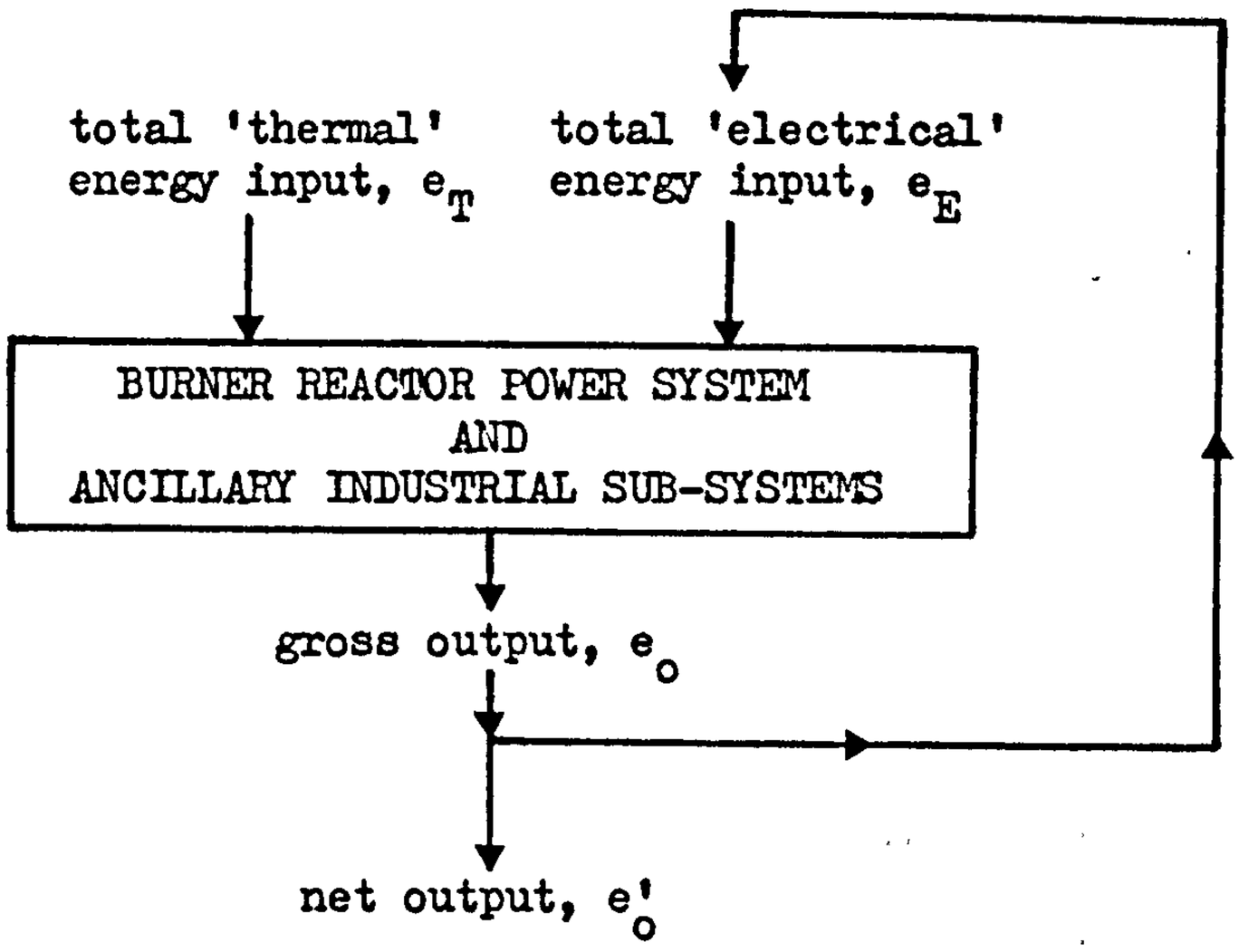
The preliminary and concurrent energy inputs, e_p and e_c respectively, consist of two general terms; an energy contribution from the consumption of electricity and an energy contribution from the use of other, 'thermal' fuels. Since all electricity used in the burner reactor power system is being provided by the system itself, any electrical energy inputs can be subtracted directly from the gross fuel output, e_o, to obtain a net fuel output, e'_o. Hence the net energy requirement, R, now equals the total thermal energy input divided by the net fuel output. This is represented diagramatically and mathematically in figure 6.4, which can be summarised by the following equation;

$$R = \text{net energy requirement of electricity}$$

$$= \frac{\text{total thermal energy input}}{(\text{gross fuel output} - \text{total electrical energy input})}$$

Formulae for net energy requirements of electricity from such partially self-supporting systems operating the primary fuel cycle were evaluated using this particular definition and the simplified energy terms e_p, e_c and e_o given in tables 6.4, 6.9

Figure 6.4 : <u>Diagram of the procedure for evaluating the net energy requirement of electricity produced by a system which consumes its own fuel.</u>



e_T = thermal energy components of the preliminary and concurrent inputs, e_p and e_c respectively.

e_E = electrical energy components of the preliminary and concurrent inputs, e_p and e_c respectively.

Net energy requirement of electricity, $R = \frac{e_T}{e'_o}$ or $\frac{e_T + e_E \times R}{e_o}$

$$= \frac{e_T}{e_o - e_E}$$

and 6.14 respectively. Equations for all six main reactor designs using fuel enriched by the gas diffusion technique are presented in table 6.17 and those for the PWR system with fuel enriched by the gas centrifuge process are shown in table 6.18. The minimum, maximum and average variation of the n.e.r., R, with uranium ore grade, G, for a PWR system operating at a load factor, l_f, of 0.62 with the primary fuel cycle including gas diffusion enriched fuel is illustrated in figure 6.5. Expressions for the average n.e.r.'s of electricity from all types of burner reactor power system are summarised in table 6.19.

These results indicate that, as the grade of ore consumed by the reactor decreases, the net energy requirement of the electricity it generates increases until it becomes infinite. At this point the system absorbs all the fuel it produces to satisfy its own electrical energy needs, that is the denominator in the equation for R equals zero. The specific ore grade at which this occurs represents a limit to the efficiency with which this type of reactor power system can 'convert' fossil fuels to electricity. This limiting ore grade for burner reactor power systems using the primary fuel cycle with gas diffusion enrichment and a load factor of 0.62 ranges from 0.5 to 70 ppm U_3O_8, with an average of 6 ppm U_3O_8.

Table 6.17 : Net energy requirements of electricity from burner reactor power systems using the primary fuel cycle, gas diffusion fuel enrichment and self-generated electrical energy.

Reactor		Net energy requirement : R (MJ[t]/MJ[e])
MAGNOX	: minimum :	$\frac{(34901_f + 11800)G + (1311_f + 20.6)}{(5663001_f - 1190)G - (40.61_f + 6.36)}$
	maximum :	$\frac{(292001_f + 23100)G + (7150l_f + 1120)}{(5662001_f - 2190)G - (5460l_f + 856)}$
AGR	: minimum :	$\frac{(4010l_f + 8970)G + (96.91_f + 12.1)}{(6169001_f - 3010)G - (29.71_f + 3.72)}$
	maximum: :	$\frac{(232001_f + 17700)G + (5270l_f + 659)}{(6098001_f - 4440)G - (4030l_f + 505)}$
SGHWR	: minimum :	$\frac{(6030l_f + 15200)G + (1311_f + 13.1)}{(8224001_f + 4170)G - (40.11_f + 4.0)}$
	maximum :	$\frac{(324001_f + 28500)G + (7130l_f + 710)}{(8125001_f - 6020)G - (5440l_f + 542)}$
CANDU	: minimum :	$\frac{(4810l_f + 28900)G + (77.71_f + 3.41)}{(8507001_f - 2270)G - (24.01_f + 1.05)}$
	maximum :	$\frac{(203001_f + 48400)G + (4230l_f + 185)}{(8503001_f - 4260)G - (3230l_f + 142)}$
BWR	: minimum :	$\frac{(5980l_f + 6590)G + (1271_f + 12.4)}{(9615001_f - 3520)G - (39.01_f + 3.78)}$
	maximum :	$\frac{(31500l_f + 16300)G + (6290l_f + 672)}{(9508001_f - 5330)G - (5290l_f + 515)}$
PWR	: minimum :	$\frac{(6530l_f + 6550)G + (148l_f + 10.0)}{(9528001_f - 3150)G - (45.51_f + 3.06)}$
	maximum :	$\frac{(360001_f + 15800)G + (8070l_f + 543)}{(9392001_f - 4850)G - (6180l_f + 415)}$

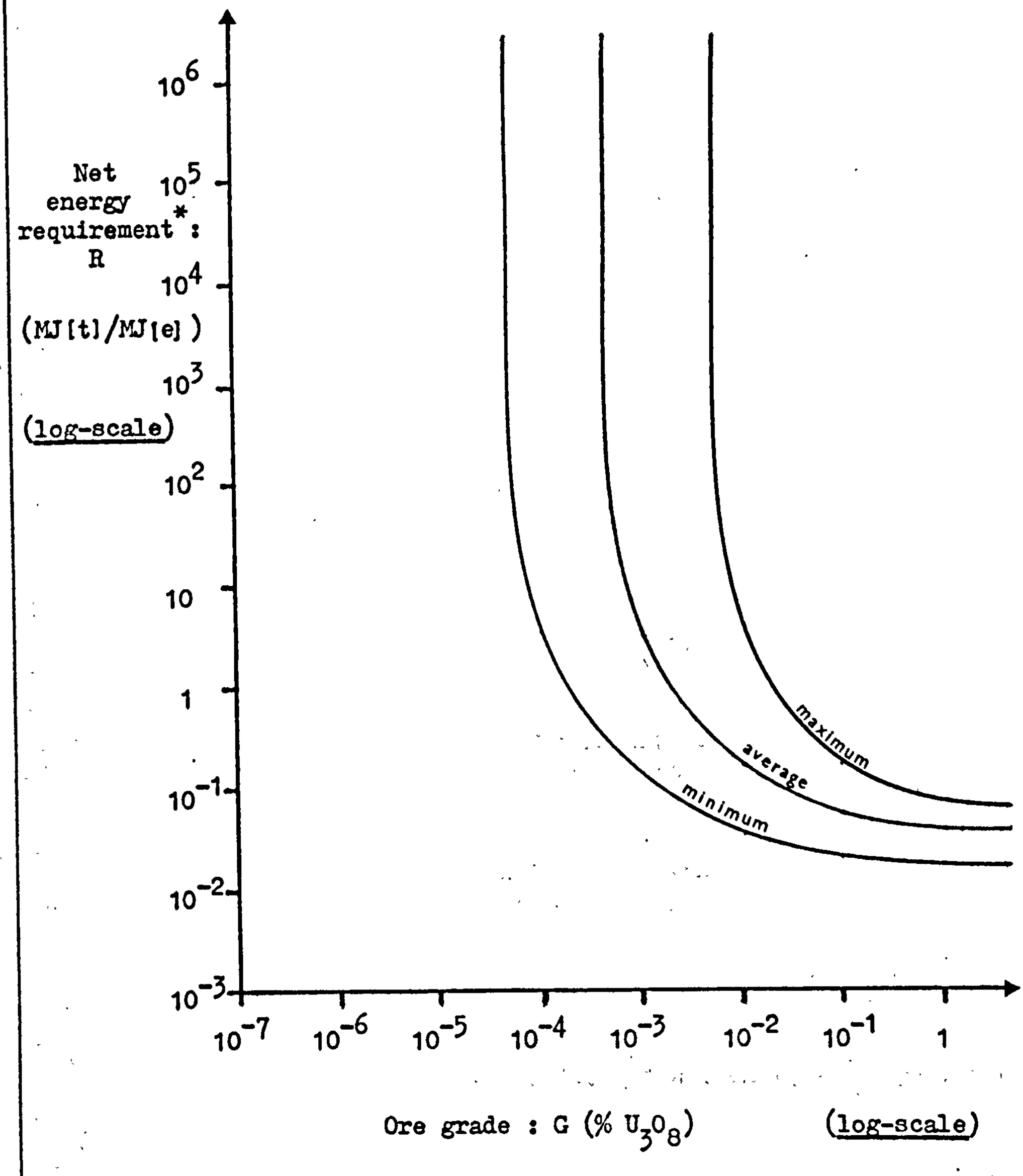
Table 6.18 : Net energy requirements of electricity from a PWR system using the primary fuel cycle, gas centrifuge fuel enrichment and self-generated electrical energy.

Value	Net energy requirement : R (MJ[t]/MJ[e])
Minimum :	$\frac{(64501_f + 6570)G + (1481_f + 10.0)}{(9896001_f - 947)G - (45.51_f + 3.06)}$
Maximum :	$\frac{(378001_f + 15900)G + (80701_f + 543)}{(9870001_f - 1990)G - (61801_f + 415)}$

Note, G = ore grade in % U_3O_8,

1_f = power station load factor.

Figure 6.5 : Ore grade variation of the net energy requirement of electricity produced by a typical burner reactor power system using the primary fuel cycle and self-generated electrical energy.*



* PWR system operating with a load factor, l_f, of 0.62, using gas diffusion fuel enrichment and consuming electricity generated by the system itself.

Table 6.19 : Average net energy requirements of electricity from burner reactor power systems using the primary fuel cycle, a load factor of 0.62 and self-generated electrical energy.

Reactor	Net energy requirement*: R (MJ[t] / MJ[e])
MAGNOX :	$\frac{27580G + 740}{349400G - 400}$
AGR :	$\frac{21770G + 530}{376600G - 250}$
SGHWR :	$\frac{33760G + 680}{501700G - 330}$
CANDU :	$\frac{46430G + 380}{524000G - 180}$
BWR :	$\frac{23060G + 610}{588400G - 270}$
PWR :	$\frac{24360G + 700}{582500G - 360}$
PWR† :	$\frac{24900G + 700}{611300G - 360}$

* assumes fuel enrichment by the gas diffusion technique except where indicated otherwise.

† assumes fuel enrichment by the gas centrifuge technique.

Note, G = ore grade in % U_3O_8.

7 PREPARATION AND USE OF BASIC RESULTS

7.1 Introduction

The purpose of this chapter is to discuss ways in which basic results obtained from the energy analysis of burner reactor power systems can be prepared for practical interpretation and application. As indicated previously, the evaluation of expressions for the net energy requirement of nuclear-generated electricity depends on certain fundamental factors such as design and operational characteristics which determine the type of system under investigation. The role of these factors must be carefully assessed in order to establish the essential framework of analysis that enables results to be used in practice. One particularly obvious application of results included here is the study of the effect of various conservation schemes on the use of fuel within a typical burner reactor power system.

7.2 Fundamental aspects of analysis

The general features of power systems which fundamentally influence the evaluation and interpretation of net energy requirement results include the power station load factor, l_f, the source of electrical energy consumed by the system and the specific design of reactor in use. The load factor is determined by the way in which the power station is operated and how it fits into the complete fuel supply system. The value of the load factor reflects the position of the generating plant in the operating schedule, or 'merit order', of the system. Power stations which occupy a high place in the merit order have large load factors since they operate almost continuously, helping to supply the majority, or 'base load', of electricity

required throughout the year. Low merit order power plant only provide electricity for daily and seasonal fluctuations in demand and hence such power stations have relatively small load factors.

The load factor affects the net energy requirement of electricity and this is demonstrated in figure 7.1 which shows average n.e.r.'s for nuclear and fossil fuel-fired power systems. The results for nuclear-generated electricity correspond to PWR systems using the primary fuel cycle with current US and South African ores (G = 0.2% and 0.05% U_3O_8, respectively), gas diffusion enrichment and self-generated electrical energy. The estimate of the n.e.r. of fossil fuel produced electricity was deduced from similar sources of data to a previous analysis of current fuel supply systems ("The energy costs of delivered energy; U.K. 1968 " Chapman, 1973b). The influence of the load factor on this results is represented by the following expression;

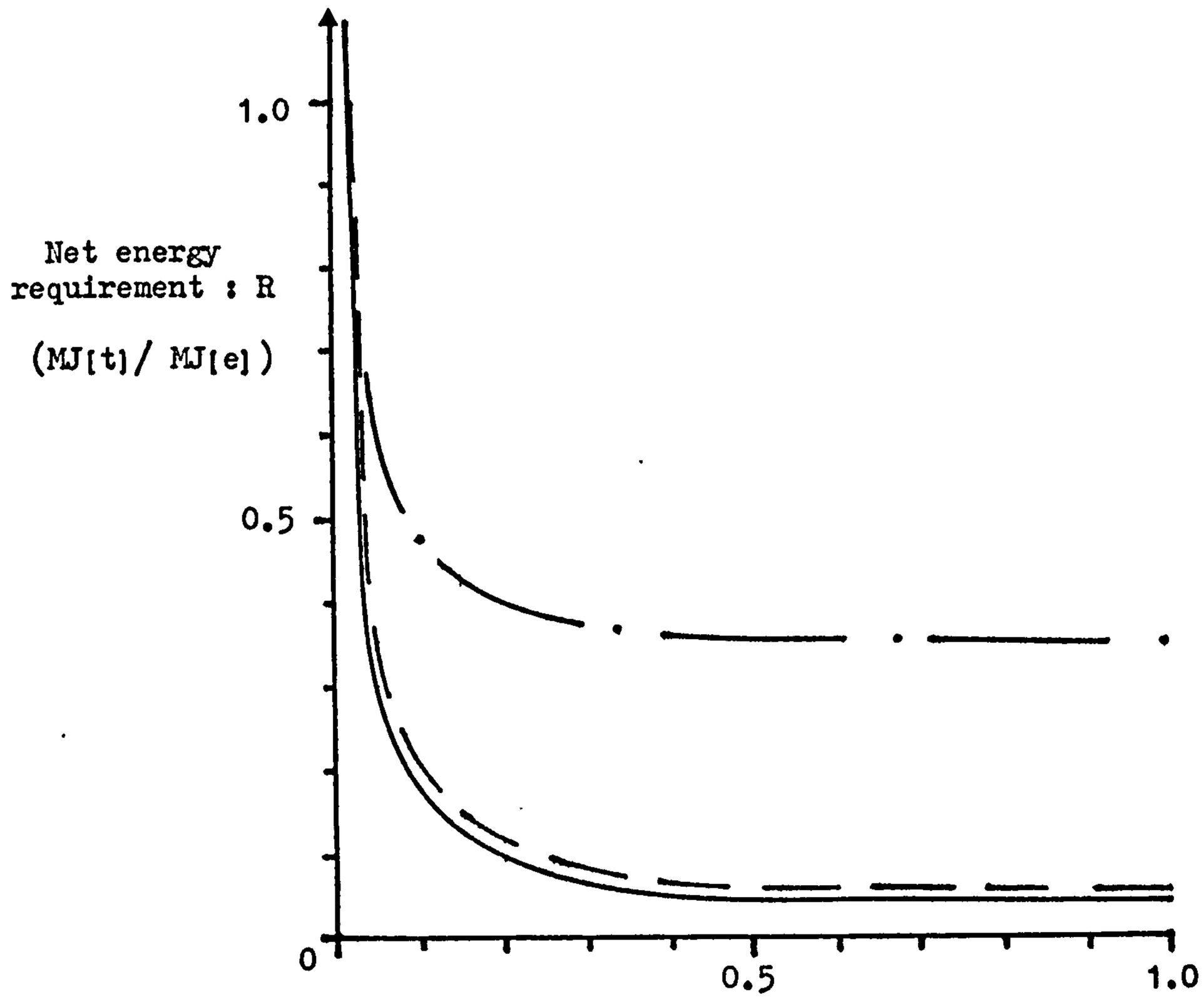
Average n.e.r. of fossil fuel-generated electricity

$$= \frac{(281.4 \times l_f) + 10.45}{(842.6 \times l_f) - 2.02} \quad MJ(t)/MJ(e)$$

where, l_f = fractional load factor of the power plant

Figure 7.1 indicates that nuclear power plant uses less fossil fuel than conventional coal-, gas- and oil-fired power stations for all values of load factor. This implies that, in energy terms, nuclear systems are more suited to continuous, or base load, operations than fossil fuel-fired systems. Assuming that this conclusion is supported by similar economic comparisons then typical load factors for nuclear base load power stations

Figure 7.1 : Load factor variation of the average net energy requirement of electricity.



——————— PWR system using current US ores (G = 0.2% U_3O_8) in a primary fuel cycle with gas diffusion enrichment and self-generated electrical energy.

—— —— PWR system using current South African ores (G = 0.05% U_3O_8) in a primary fuel cycle with gas diffusion enrichment and self-generated electrical energy.

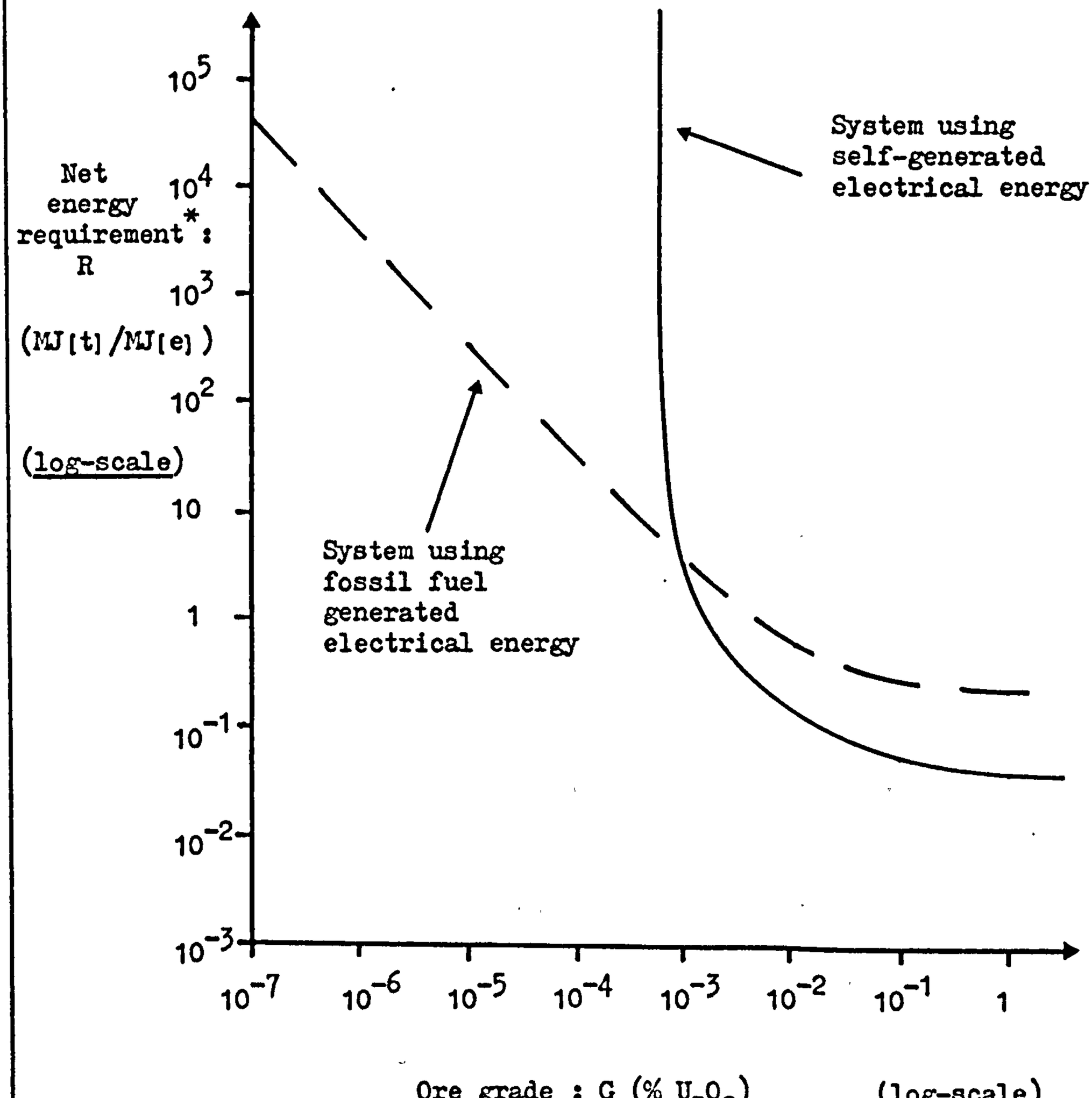
—— • —— Self-supporting fossil fuel-fired power system.

seem to be the most appropriate to use in any further analysis. Consequently an average lifetime value of the load factor of 0.62 (Searby, 1971) was used in the following investigations.

The specific source of electrical energy consumed by burner reactor power systems obviously affects the net energy requirement of the electricity they produce. This was indicated previously (cf. figures 6.3 and 6.5) and it is additionally demonstrated in figure 7.2 which shows the average n.e.r.'s of electricity for typical systems using both externally and internally produced electrical energy. Differences between the two estimates are caused by the fact that the system supplied with electrical energy from other, independent systems, such as fossil fuel-fired power plant, avoids any cyclic interrelationship, or feedback, between the fuel output and the energy input. Consequently, by using electricity obtained from processes with an artificially fixed electrical conversion factor of 4 MJ(t) per MJ(e), the reactor power system can operate, in some circumstances, more efficiently ($G > 0.001\% U_3O_8$) and, in others, less efficiently ($G < 0.001\% U_3O_8$), than the system that initially supplies this electrical energy. Since this is a rather unrealistic situation, an automatically adjusted electrical conversion factor is incorporated into the following analysis by mainly studying reactor systems which use self-generated electrical energy.

The average net energy requirement of electricity produced by different burner reactor power systems may now be evaluated and compared using these basic principles. Results for systems incorporating the MAGNOX, Advanced Gas-cooled Reactor (AGR), Steam Generating Heavy Water Reactor (SGHWR), CANDU, Boiling

Figure 7.2 : Comparison of the average net energy requirement of electricity from burner reactor power stations using self-generated electrical energy and electrical energy from fossil fuel-fired power plant.*



* PWR systems operating with a load factor of 0.62 on the primary fuel cycle incorporating fuel enriched by the gas diffusion technique.

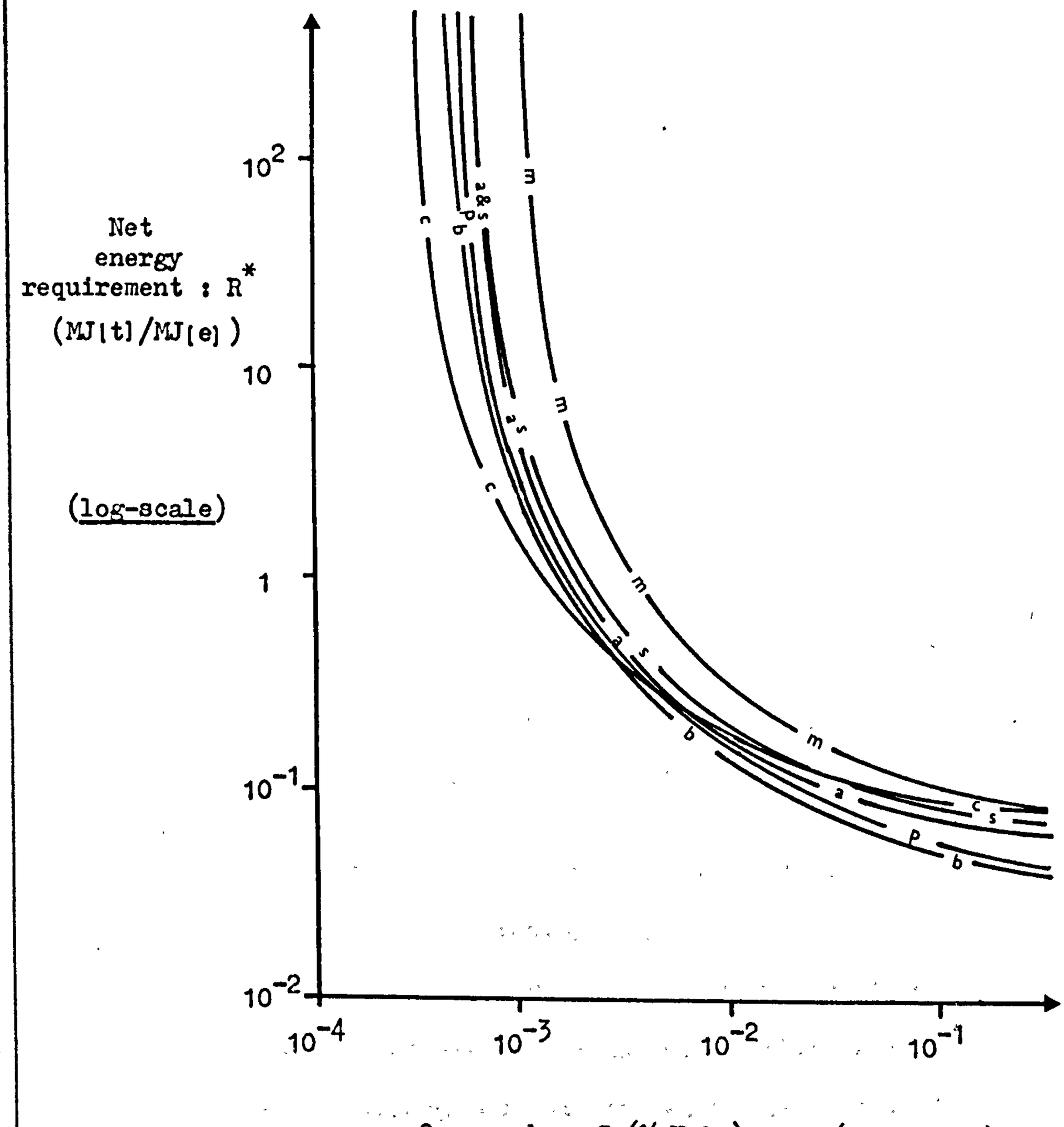
Water Reactor (BWR) and Pressurised Water Reactor (PWR) designs are illustrated in figure 7.3. This shows that n.e.r.'s for all these systems are quite similar. The system based on the MAGNOX reactor produces slightly higher n.e.r. values than the rest for all ore grades, whilst the lowest values correspond to the BWR system using relatively rich ores and the CANDU system with poorer ores. The figure indicates that, on an energy basis, the PWR system can be regarded as a fairly typical example of current burner reactor power systems.

Hence in most of the applications of energy analysis presented in this particular study a standard type of burner reactor power system will be investigated. This system will basically consist of a typical PWR plant operating at an average lifetime load factor of 0.62. The specific source of electrical energy used by the system will depend on the actual context of analysis. If the system is being compared with the current situation, then it will be assumed that all electrical energy is supplied by fossil fuel-fired generating plant with a typical conversion factor of 4 MJ(t) per MJ(e). In cases where the absolute efficiency of the system is being assessed it will be necessary to assume that the system itself provides all the electrical energy it consumes. These fundamental frameworks enable results to be interpreted correctly and used on a qualified basis.

7.3 Energy analysis and fuel conservation

Many different processes contribute to the production of electricity from uranium by burner reactor power systems. The proportion of the energy consumed in various parts of the fuel cycle, and of the construction and operation of the power

Figure 7.3 : Comparison of the average energy requirement of electricity from various burner reactor power systems using the primary fuel cycle and self-generated electrical energy*.



* assumes a load factor of 0.62 and fuel enrichment by the gas diffusion technique.

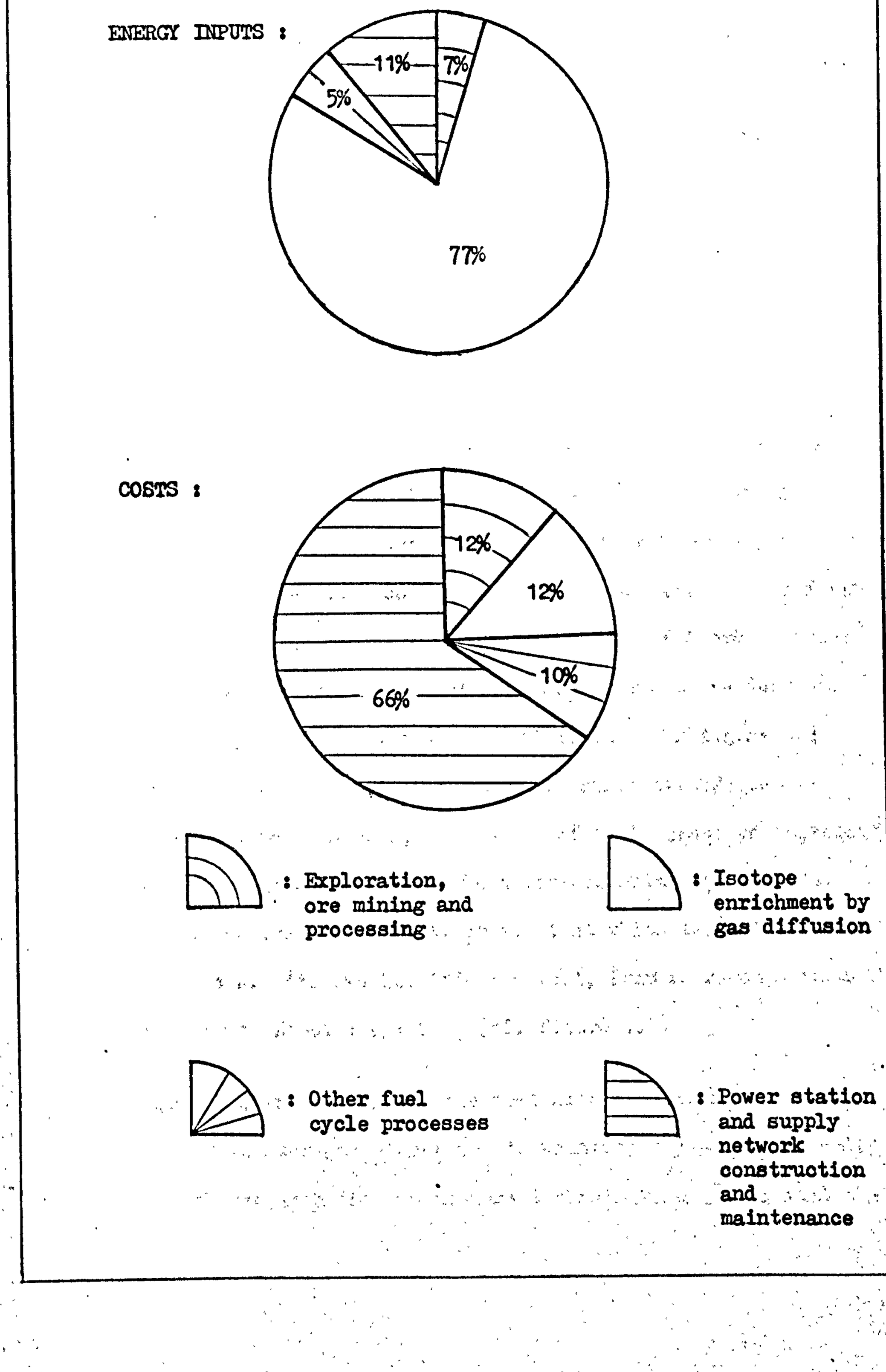
station and electricity supply network can be deduced from the results presented in chapters 5 and 6. A breakdown of the energy contributions of uranium concentrate production, uranium isotope enrichment, other fuel cycle operations, and the construction and maintenance of the power station and transmission and distribution system is illustrated in figure 7.4.

This diagram refers to a typical PWR system operating at a load factor of 0.62 and using a primary fuel cycle which incorporates gas diffusion enrichment and a current ore of grade 0.2% U_3O_8. This particular analysis of fuel use was obtained assuming a currently applicable conversion factor for electrical energy of 4 MJ(t) per MJ(e) and the results may be compared with the proportional contributions to the cost of nuclear-generated electricity (Organisation for Economic Co-operation and Development, 1974; Sir John Hill, 1974). These economic data correspond to an estimated total cost of electricity of about 4×10^{-3} dollars (US) per mega-joule, that is 4 mills/MJ(e) or approximately 0.6 pence/kWhe (1974 values).

The energy contributions shown in figure 7.4 indicate the main areas where significant savings may be achieved by fuel conservation measures. The relative importance of these schemes can be assessed by evaluating the changes in the net energy requirement of nuclear-generated electricity with various measures such as the use of new ore mining and processing techniques, improved isotope enrichment methods and different fuel management policies.

Although the contribution of ore mining and processing to the total energy input of producing electricity from burner reactor

Figure 7.4 : Breakdown of the total energy input and cost of electricity produced by a typical burner reactor power system.

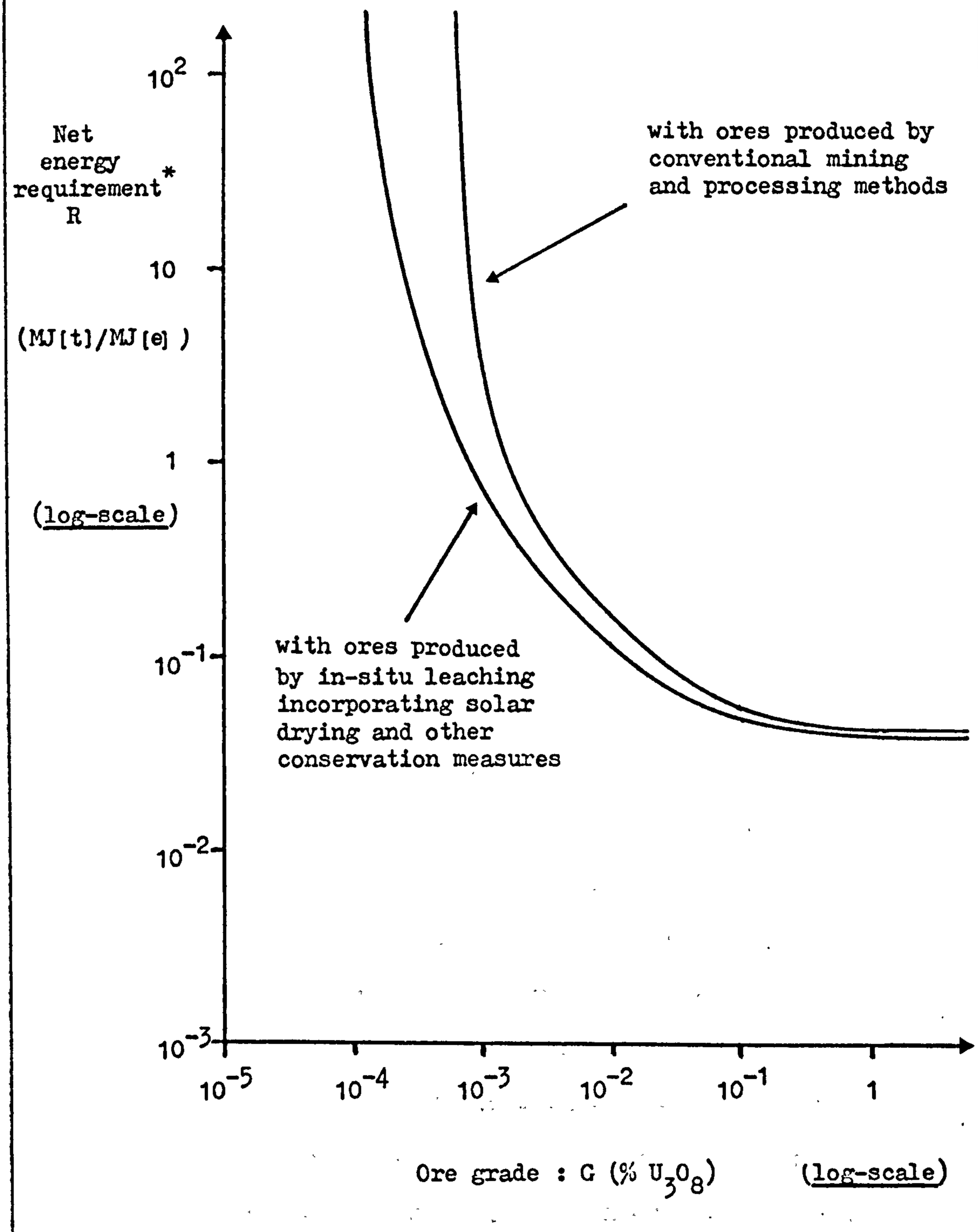


power systems is relatively small, about 7% for current US ores (G = 0.2% U_3O_8), this proportion rises dramatically with falling ore grades. For example, the energy input of producing uranium concentrate from typical South Afican ores (G = 0.05% U_3O_8) accounts for approximately one quarter of the total input. At the moment mining and comminution contribute roughly a third of the energy requirement of concentrate production, whilst other ore processing operations such as leaching, uranium recovery and concentrate drying are responsible for the remainder.

Fuel consumption in this part of the nuclear fuel cycle could be reduced by the use of thermal insulation, waste heat and solar energy. If present technical difficulties could be overcome then new ore mining and processing methods such as in-situ leaching, which involves recovering uranium-rich liquors directly from ore deposits whilst avoiding energy intensive excavation, comminution, etc., operations, could also be introduced to conserve fuel. The effect of these measures on the net energy requirement of electricity can be deduced from energy analysis and the result is shown in figure 7.5. This demonstrates that, although only small savings can be achieved with rich ores, the amount of fuel conserved increases as the ore grade declines. Such improvements can, in fact, reduce the ore grade limit, or point at which the system consumes all its own electrical output, from an average value of 6 ppm to about 1 ppm U_3O_8 (cf. figure 6.5).

Figure 7.4 indicates that the most significant part of the fuel cycle as regards fuel use is uranium isotope enrichment. Initial research into enrichment technology suggested that the

Figure 7.5 : Comparison of the average net energy requirement of electricity from a typical burner reactor power system using different ore mining and processing methods.*

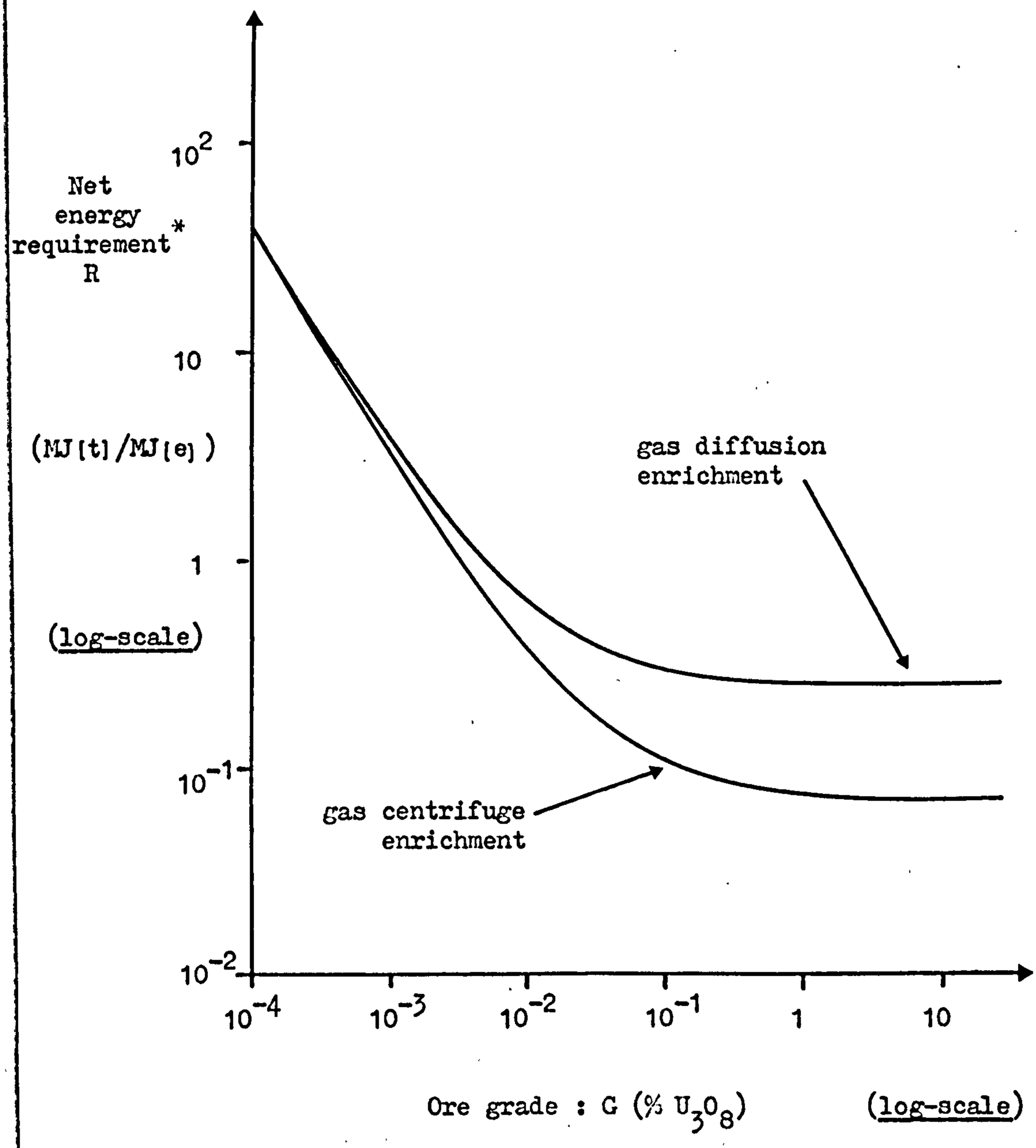


* for a PWR system assuming all electrical energy is supplied by the system itself, gas diffusion enrichment is used, the primary fuel cycle is in operation and the load factor is 0.62.

energy required to produce one unit of enrichment could fall to a tenth of the present level if gas centrifuge techniques currently under development were to replace the widespread gas diffusion process (see appendix H). The current effect on fuel consumption can be best demonstrated by comparing n.e.r.'s of electricity produced from systems which incorporate either of these enrichment methods and consume electrical energy from fossil fuel-fired power plant, that is with an electrical conversion factor of 4 MJ(t) per MJ(e). The results are given in figure 7.6 which shows that savings of between two thirds and three quarters can be achieved with systems using rich ores. However, as ore grades decrease the energy input of concentrate processing begins to dominate the n.e.r. and improvement in enrichment methods has no effect on the energy criterion which determines the lowest grade of ore that can be used effectively by the system.

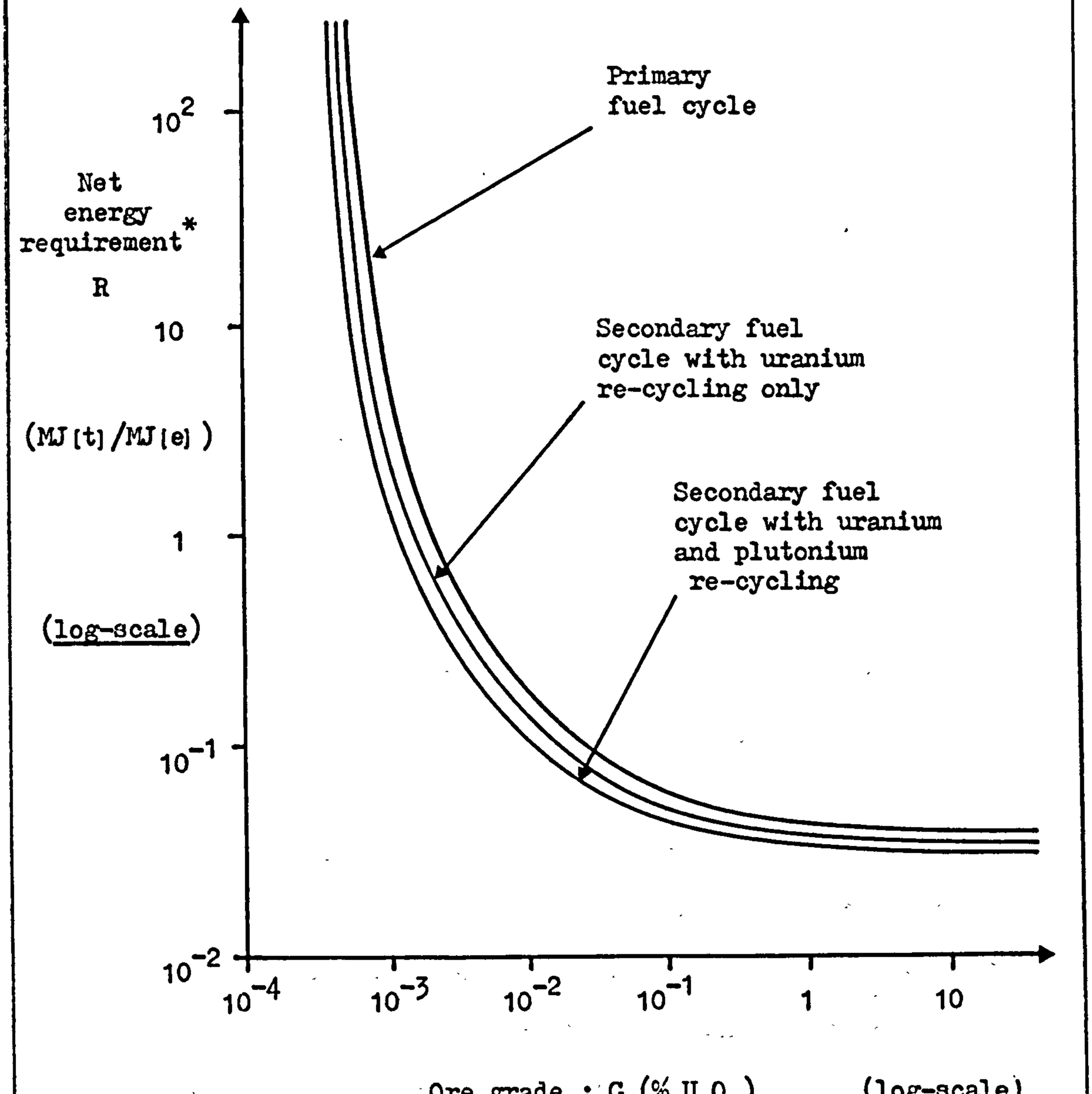
At the moment the amount of energy required to produce nuclear fuel from naturally-occurring uranium (N_A) is slightly more than that needed to recover and re-cycle uranium from spent fuel (N_B) and considerably more than that used to obtain plutonium from spent fuel (N_C). Since these differences increase as ore grades fall, the use of re-cycled, or secondary-produced, material should introduce significant fuel savings. The effect of various fuel management policies on energy consumption are considered here by comparing the n.e.r.'s of electricity from systems which use the primary fuel cycle, single-stage uranium re-cycling and uranium/plutonium mixed oxide re-cycling (see 6.3). Figure 7.7 illustrates that as much as a fifth of the present total fuel input can be conserved by such measures.

Figure 7.6 : Comparison of the average net energy requirement of electricity from a typical burner reactor power system using different uranium isotope enrichment techniques.*



* for a PWR system using a primary fuel cycle, operating at a load factor of 0.62 and consuming electrical energy provided by fossil fuel-fired power plant.

Figure 7.7 : Comparison of the average net energy requirement of electricity from a typical burner reactor power system using different fuel management policies.*



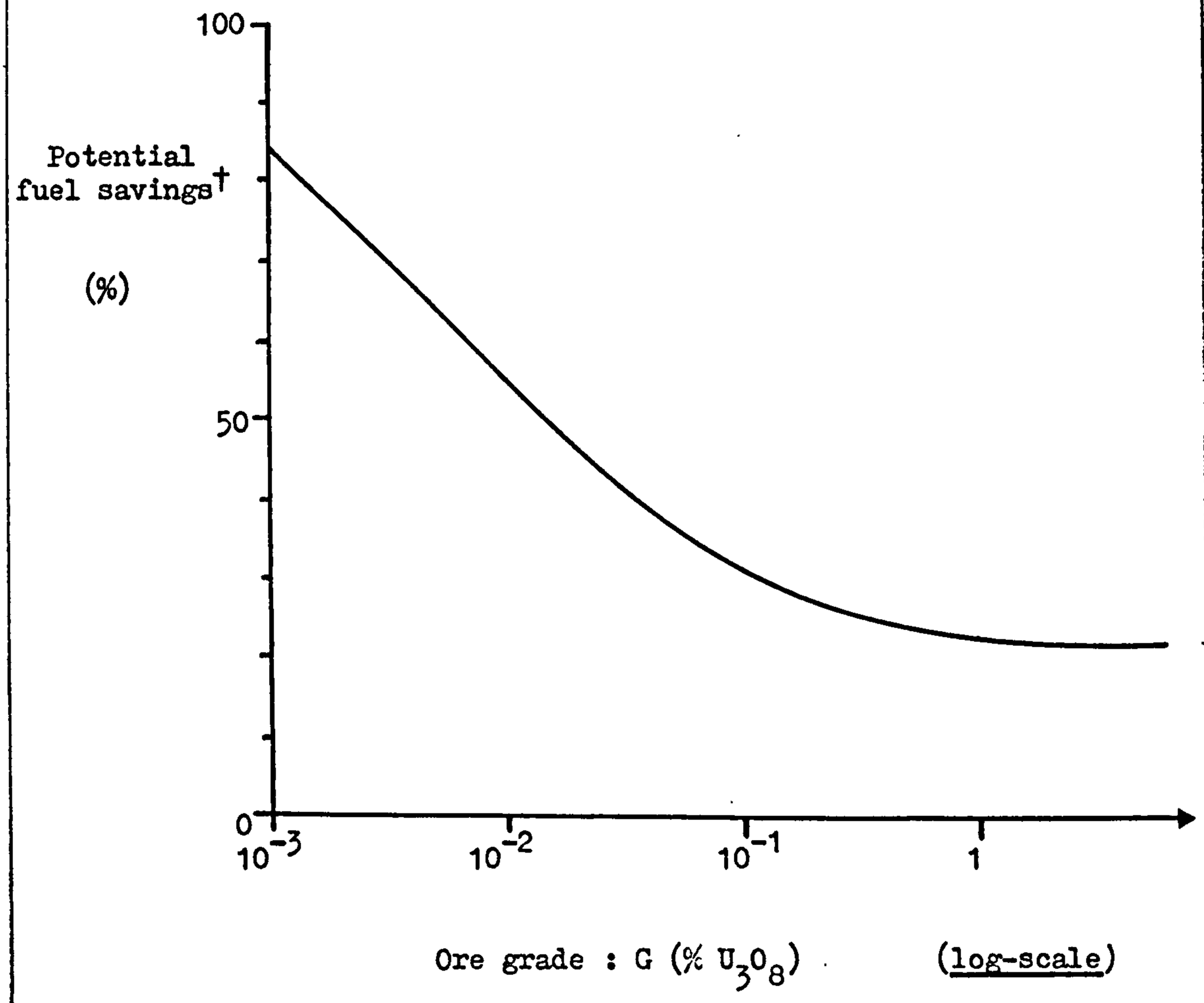
* for a PWR system using gas diffusion enrichment techniques, consuming electrical energy provided by the system itself and operating at a load factor of 0.62.

The total fuel savings that can be achieved by combining all these conservation measures are indicated in figure 7.8. This shows the difference between the amount of fuel used by a PWR system incorporating a mixed oxide secondary fuel cycle with ore mining and processing improvements and gas centrifuge enrichment and the quantity consumed by a standard system operating on the primary fuel cycle with conventional mining and ore processing and gas diffusion enrichment. The savings can be substantial and rise as ore grades decline. In addition the ore grade limit, defined as the point at which the system consumes all its own electrical output, is reduced by the conservation schemes from 6 ppm to about 0.8 ppm U_3O_8.

Although these potential savings may initially seem very attractive, a number of important technical and economic considerations must first be satisfied before they can be successfully achieved. Since direct fuel conservation by insulation and the use of waste heat from power plants and other machinery usually involves significant capital investment which may take many years to repay (eg. Lambert, 1973; Building Research Establishment, 1975; New Scientist, 1977), immediate advantages for individual firms and companies are not often obvious. Solar energy can currently be used to provide low-grade heat and subsequently reduce fuel consumption, but geographical, climatological and seasonal factors can restrict large scale industrial applications.

At the moment technical and geological factors limit the widespread use of in-situ leaching (Mining Magazine, 1971b), whilst slow commercial development delays the industrial operation of new gas centrifuge enrichment capacity (Roberts, 1973). Re-cycling nuclear fuel requires extensive reprocessing

Figure 7.8 : Potential fuel savings for a typical burner reactor power system.*



* for a PWR system operating at a load factor of 0.62 and consuming self-generated electrical energy.

† Standard system uses conventional ore mining and processing methods, gas diffusion enrichment and the primary fuel cycle. New system incorporates fuel conservation measures such as in-situ leaching, solar drying, gas centrifuge enrichment and the secondary fuel cycle with uranium and plutonium re-cycling.

facilities which have yet to be designed, constructed and operated reliably (Rippon, 1976). The introduction of re-cycled uranium and plutonium into burner reactor fuel cycles can incur engineering problems (Resnikoff, 1975; Puechl, 1975) and other significant drawbacks that have been suggested involve nuclear weapons proliferation, environmental hazards, public safety and threats to civil liberties (eg. Grant, 1977; Wright, 1977).

Even assuming all difficulties concerning the implimentation of fuel conservation measures can be quickly solved or avoided, the sort of savings demonstrated in figure 7.8 could not be achieved immediately. A certain amount of time is required to incorporate improvements and build new plants. The period between considering and commissioning nuclear facilities can easily amount to ten years, whilst actual construction times for single new mines and mills can range from three to four years (Youngberg, 1973; Fishlock, 1975), for new enrichment capacity from three to five years (Mohrhauer, 1972) and for reprocessing plant approximately five years (Rippon, 1976). In addition suitable incentives are needed to encourage improvements in the nuclear power system and it can be seen from figure 7.8 that greater savings cannot be realised until ores of much lower grades than those currently worked are used. Consequently attempts to conserve fuel in the nuclear industry may well be delayed for many years.

8 FURTHER APPLICATION OF RESULTS

8.1 Introduction

In addition to forming the basis for investigating the savings from fuel conservation, the results of this energy analysis can also be used to estimate the total amount of uranium from resources that can be utilised effectively by burner reactor power systems. As demonstrated in chapter 6 the variation of the net energy requirement of nuclear-generated electricity with uranium ore grade implies that there are restrictions on the quality of resources which can be consumed efficiently by such systems. In this chapter the evaluation of such ore grade limits is studied in more detail. Wider implications are examined by using results to formulate a simple economic model which relates the cost of producing electricity from nuclear sources to uranium ore grade. Subsequently constraints on the quantity of economically recoverable uranium are then investigated.

8.2 Energy analysis and ore grade limits

The amount of energy required to produce electricity from burner reactor power systems depends on many factors not least of which is the uranium ore grade, G. The ore grade indicates the relative quantity of potentially useful material contained in a naturally-occurring resource and for uranium this is measured in terms of the percentage of triuranium octoxide (U_3O_8) in the ore. However, for convenience, the uranium content of an ore can be written in parts per million (ppm : 10^{-6}) or parts per billion (ppb : 10^{-9}) U_3O_8.

The net energy requirement of nuclear-generated electricity indicates how efficiently burner reactor power systems use fuel and other, non-nuclear energy resources. Hence, the variation of this n.e.r. with uranium ore grade can be used in conjunction with certain energy criteria to identify those grades for which the system no longer functions effectively. Specific points at which this occurs are referred to here as ore grade limits and estimates of these parameters for a typical PWR system operating under various fundamental conditions are shown in table 8.1.

In this table the 'fuel-dependent' case refers to a system that obtains all the fuel it uses from other, quite different fuel supply industries. This corresponds to the current situation in many countries where the majority of electricity available for consumption is generated by fossil fuel-fired power plant. Consequently the conversion factor for electrical energy applied in this case is 4 MJ(t) per MJ(e). The ore grade limit shown is defined by the somewhat arbitrary condition that the n.e.r. of nuclear-generated electricity equals the g.e.r. of fossil fuel-fired electricity (i.e. R = 4 MJ(t)/MJ(e), see figure 6.3). When this limit is reached the nuclear system is only as efficient as conventional plant at 'converting' fossil fuels into electricity.

The 'electricity-independent' case in table 8.1 indicates the ore grade limit for a nuclear system that consumes self-generated electrical energy and relies on other industries for all other fuels. This case illustrates the effect of producing all electricity from a totally nuclear-powered electricity supply system. The ore grade limit in this situation is encountered when the output of the system equals

Table 8.1 : Ore grade limits for a typical burner reactor power system.*

Type of system	Ore grade limit (ppm U_3O_8)		
	minimum	maximum	average
Fuel-dependent :	1.0	100	10
Electricity-independent :	0.5	70	6
Fuel-independent :	1.5	150	15

* PWR system operating at a load factor of 0.62 and using the primary fuel cycle with gas diffusion enrichment.

its own electrical energy input (i.e. R =∞, see figures 6.5 and 7.7).

The 'fuel-independent' case corresponds to a situation where the nuclear system produces all its own fuel. This special case represents what would happen in an 'all electric, all nuclear' economy where the only fuel is electricity that is supplied by a totally nuclear-powered fuel system. The ore grade limit is determined by the critereon that the system consumes all its own output as fuel. This limit is deduced with n.e.r.'s calculated from the results of chapter 6.

The net energy requirement of electricity produced by a 'fuel-independent', or fully self-supporting, burner reactor power system is evaluated by translating the previously calculated estimates of coal, gas, oil, etc., consumption, or 'thermal' energy inputs, into terms of electrical energy. To achieve this a conversion factor, j, is required to show the amount of electricity that can substitute one unit of gross thermal energy. Since thermal energy input figures indicate the total amount of energy resources extracted by any given fuel consumption, the substitution factor, j, must incorporate efficiencies of the production and delivery of fuels, n_a, as well as the efficiency of their use, n_b. The efficiency of the use of electricity, n_c, must also be included in this substitution factor which is represented by the expression;

j = amount of electricity used to replace one unit of any other type of fuel

= electrical input per unit gross thermal energy input

= substitution factor

$$= n_a \times \frac{n_b}{n_c} \qquad \text{MJ(e)/MJ(t)}$$

where,

n_a = efficiency of the production and delivery of 'thermal' fuels

= inverse gross energy requirement of fuel in mega-joules

n_b = efficiency of 'thermal' fuels at the point of use

n_c = efficiency of electricity at the point of use

Typical values of the production efficiency, n_a, for various common fuels are given in table 8.2. These figures correspond to the g.e.r. results used throughout this study which are also presented in table 3.1. The 'point of use' efficiency factors for thermal energy, n_b, and electrical energy, n_c, are illustrated in table 8.3. By combining these estimates together a typical substitution factor can be deduced;

$$j = 0.65 \pm 0.05 \quad MJ(e)/MJ(t)$$

Net energy requirement estimates for systems which produce all the fuel they need can be found by using this factor in conjunction with previous values of the energy inputs e_p and e_c and the fuel output e_o shown in tables 6.4, 6.9, 6.11, 6.13 and 6.14. Minimum and maximum n.e.r. results for a typical PWR system using the primary fuel cycle with gas diffusion enrichment are illustrated in table 8.4 and variations with ore grade are demonstrated in figure 8.1. Average values for systems using different fuel management policies and enrichment techniques are presented in table 8.5.

The ore grade limit for this type of self-supporting nuclear system is defined by the condition that the total fuel output equals the total fuel input, that is R = 1 MJ(e) per MJ(e). As indicated in table 8.1 the ore grade limit for this case ranges from 1.5 ppm to 150 ppm, with an average of 15 ppm U_3O_8.

Table 8.2 : Typical fuel production and delivery efficiency factors

(from Chapman, 1973b)

Fuel	Efficiency : n_a^*
Coal	0.96
Coke	0.85
Natural gas	0.94
Oil	0.88

* Net fuel delivered to the point of use per gross energy input.

Table 8.3 : Typical 'point of use' efficiency factors (based on data from; National Economic Development Office, 1974; Building Research Establishment, 1975; Handbook of Electricity Supply Statistics, 1976).

Industrial application	Efficiency* n_b	n_c
Space heating	0.70 ± 0.10	0.975 ± 0.025
Water heating	0.75 ± 0.05	0.95 ± 0.05
Furnace heating	~0.55	~0.95
Drying	~0.45	~0.55
Transport	0.15 ± 0.05	0.25 ± 0.10†

* useful energy output per direct fuel energy input.

† batteries and fuel cells for road transport, electric locomotives for rail transport.

Table 8.4 : Net energy requirements for a typical burner reactor power system which is completely self-sufficient in fuel.*

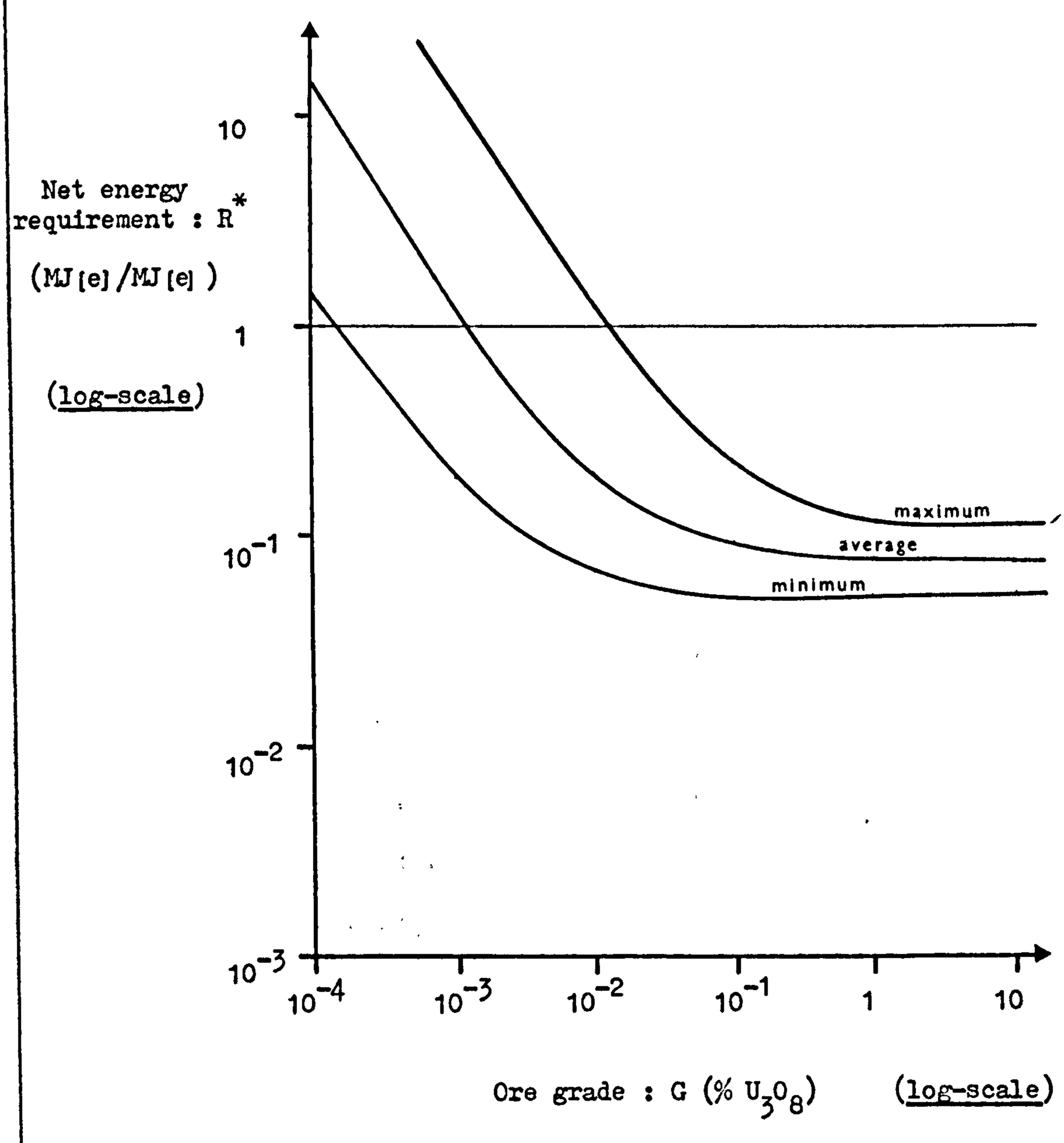
Value	Net energy requirement : R (MJ[e]/MJ[e])
Minimum :	$\frac{(9.1\times10^{-6}}{G} + 0.007)/_{l_f} + \frac{(1.3\times10^{-4}}{G} + 0.045)$
Maximum :	$\frac{(8.0\times10^{-4}}{G} + 0.016)/_{l_f} + \frac{(0.012}{G} + 0.080)$

* PWR system using the primary fuel cycle with gas diffusion enrichment.

Note, G = ore grade in % U_3O_8

l_f = load factor.

Figure 8.1 : Ore grade variation of the net energy requirement of electricity produced by a typical burner reactor power system which is completely self-sufficient in fuel.*



* PWR system using the primary fuel cycle with gas diffusion enrichment.

Table 8.5 : Average net energy requirement of electricity from typical burner reactor power systems, which are completely self-sufficient in fuel, using different fuel management policies.

Fuel management policy	Net energy requirement*: R (MJ[e]/MJ[e])
Primary fuel cycle :	$\frac{0.0013}{G} + 0.0810$
Secondary fuel cycle with uranium re-cycling:	$\frac{0.0010}{G} + 0.0795$
Secondary fuel cycle with uranium and plutonium re-cycling :	$\frac{0.00085}{G} + 0.0668$
Secondary fuel cycle with uranium and plutonium re-cycling† :	$\frac{0.00085}{G} + 0.0293$

* assumes gas diffusion fuel enrichment except where indicated otherwise.

† assumes gas centrifuge fuel enrichment.

Note, G = ore grade in % U_3O_8.

These ore grade values could be regarded as ultimate limits to the quality of resources that can be used effectively by the system, since with poorer ores the system would consume more energy in the form of electricity than it is capable of producing. In this situation the system would be totally impractical as a producer of fuel.

Although this sort of analysis can reveal physical limitations to the successful operation of burner reactor power systems, the relative value of ores and the proportion of uranium resources which are actually available for use are determined by economic considerations. Figure 8.1 shows that as the ore grade approaches the defined limit the amount of fuel which carries the full burden of system costs falls rapidly until it equals zero. This implies that the cost of nuclear-generated electricity will rise dramatically and, since the profitability of the system is governed by relative rather than absolute costs, the actual ore grade limit will be attained well in advance of the limit defined purely by energy criteria. Hence, any attempt to evaluate uranium resources for burner reactor power systems must be based on some form of economic assessment of nuclear power costs.

8.3 Nuclear power costs

The economics of generating electricity from nuclear fission burner reactor power systems has been studied by numerous researchers for many different reasons. Some have investigated the effect of technological factors on costs (eg. Bader, Kitzke and Nordman, 1969) and others have examined the role of capital interest rates, amortisation, inflation, etc.(eg. Berrie and Betts, 1967; Akhtar, 1974; Bupp, Derian, Donsimoni and Treitel,

1975). The purpose of this particular investigation is to formulate a very simple economic model based primarily on the results of energy analysis. In particular, the nature of this model will enable the influence of such factors as uranium ore grade on the cost of nuclear-generated electricity to be studied in some detail.

In basic terms the total cost of any item or service equals the sum of all cost components associated with its production and supply. The basic assumption incorporated into this simple economic model is that such cost components can be divided into one of two distinctly different categories; fuel costs and other, or 'non-fuel', costs. Fuel costs are the product of fuel consumption and fuel prices, and such costs are incurred by all activities which introduce either direct or indirect energy inputs to the system. Other, or non-fuel, costs include such items as wages, interest, etc., that are paid for inputs such as manpower, capital, etc. By definition, non-fuel costs contain no element, however obscure, of fuel use. The purpose of segregating costs which are determined by the price of fuels from those which are not is to enable the economic effect of energy feedback mechanisms within the system to be examined.

Energy feedback, or interrelationship between the amount of fuel a system produces and the quantity it consumes, can fundamentally influence the cost of fuels. For a self-supporting system, fuel costs alone can dictate the sensitivity of the total cost to basic changes in the system such as, for example, the use of poorer ores in a burner reactor power system. The importance of fuel costs prevails despite the contribution of non-fuel costs which can, in some circumstances, be comparatively large. Fuel costs equal the

product of fuel consumption and the internal fuel price, whilst other costs depend on the level of externally determined prices. Hence fuel costs are the most essential component of the total cost since an increase in the price of electricity immediately causes fuel costs to rise but has little or no impact on non-fuel costs.*

The fundamental cost of nuclear-generated electricity, c_e, can be expressed in terms of the net energy requirement of electricity, R, the price of electricity, p_e, and non-fuel costs, c_o, by the following equation;

c_e = fundamental cost of nuclear-generated electricity

$= (R \times p_e) + c_o$

where,

R = n.e.r. of nuclear-generated electricity

p_e = price of nuclear electricity

c_o = cost of non-fuel inputs to the production of nuclear electricity

assuming $p_e = c_e$ then,

$$c_e = \frac{c_o}{1 - R}$$

---

* The price of electricity can be linked to the level of some non-fuel costs such as wages. However this connection is generally less effective and often more vague than the relationship between fuel costs and fuel prices. Hence this effect is not incorporated into this simple model.

The assumption that the price of electricity equals its cost seems fairly reasonable for a system which supplies its own fuel needs. An approximation to this should also hold in cases where the system obtains fuel from other industries, since intercompetition between fuels can rarely support large price disparities.

Values of the n.e.r. of electricity, R, from a wholly self-supporting system are used in the above equation and results, which were evaluated earlier (see 8.2), are presented in tables 8.4 and 8.5. The total non-fuel cost of producing electricity, c_o, equals the sum of non-fuel cost components of each input to the system. Estimates of these components were deduced by subtracting the fuel cost element of each input from its current (1965 - 70) total cost. Data on the total cost of inputs such as fuel cycle processes, power plant construction and system operation were obtained from various economic studies (eg. Engineering and Mining Journal, 1961; Bader et al, 1969; Lankenau and Light, 1971; Youngberg, 1973; Akhtar, 1974; Bupp et al, 1975) and typical results are given in table 8.6.

Fuel cost components for these inputs were calculated from fuel consumption figures deduced by energy analysis and values of current (1965 - 70) fuel prices. The fuel prices used consisted of 2.5 ± 0.5 mills per MJ(e) for electrical energy and 0.35 ± 0.10 mills per MJ(t) for thermal energy from coal, fuel oil, natural gas, etc. (Report on the Census of Production, 1968; Statistical Abstract of the United States, 1975; Handbook of Electricity Supply Statistics, 1976). Fuel cost components and subsequent non-fuel cost contributions are also shown in table 8.6.

Table 8.6 : Average estimates of typical PWR system unit costs (1965 - 1970 values).

Input	Unit	Fuel cost	Non-fuel cost	Total cost
		($ per unit)		
Exploration	kg U_3O_8	0.02	1.08	1.10
Mining	te ore	1	12	13
Ore processing	te ore	1	5	6
Refining and conversion	kg U	0.2	1.7	1.9
Enrichment				
- gas diffusion	kg S.W.U.	26	9	35
- gas centrifuge	kg S.W.U.	5	30	35
Fuel fabrication	kg U	1	59	60
Reprocessing	kg U	0.4	22.6	23
Waste disposal*	kg U	0.1	10.6	10.7
Power plant construction†	kW(e)	5	165	170
Power plant labour	MJ(e)	-	3×10^{-4}	3×10^{-4}
Transmission and distribution	MJ(e)	-	1×10^{-3}	1×10^{-3}

* glassification and burial.

† excluding the initial core.

The total non-fuel cost of producing electricity from burner reactor power systems, c_o, can be deduced from this information in much the same way as net energy requirement results were calculated from basic energy terms in chapters 5 and 6. Table 8.7 illustrates average values of these costs for a PWR system operating at a load factor of 0.62 and using the type of nuclear fuel described previously in tables 5.16 and 5.18. Equations for the fundamental cost of nuclear-generated electricity, c_e, were obtained by combining these estimates with n.e.r.'s for electricity from tables 8.5 and 8.6 and subsequent final results are presented in table 8.8.

These expressions for the fundamental cost of electricity, c_e, can be used to assess the effect of basic variables on the economics of nuclear power systems. The full variation of the cost of electricity with ore grade is shown in figure 8.2 for a typical PWR system operating at a load factor of 0.62 and using the primary fuel cycle with gas diffusion enrichment. This figure indicates that, for current ore grades (G = 0.2% U_3O_8), the cost of nuclear-generated electricity should be between 1.3 and 3.0 mills per MJ(e). This may be compared with actual total costs of 2.0 mills per MJ(e) (based on 1965 - 70 data from; Central Electricity Generating Board, 1971; Searby, 1971; Handbook of Electricity Supply Statistics, 1976).

According to figure 8.2 the total cost of nuclear-generated electricity is fairly constant over the wide range of high grade ores (on average, G > 0.1% U_3O_8) that have been the traditional sources of uranium. This result reflects the observed fact that, discounting the effects of general inflation, the relative cost of nuclear-generated electricity has been reasonably stable over a number of years (Handbook of

Table 8.7 : Average estimates of the total non-fuel cost of electricity from a typical PWR system (1965 - 1970 values).

Fuel management policy	Total non-fuel cost*: c_o (mills/MJ[e])
Primary fuel cycle†	$\frac{0.0155}{G} + 1.83$
Secondary fuel cycle with uranium re-cycling†:	$\frac{0.0133}{G} + 1.84$
Secondary fuel cycle with uranium and plutonium re-cycling :	$\frac{0.0104}{G} + 1.83$
Secondary fuel cycle with uranium and plutonium re-cycling‡ :	$\frac{0.0104}{G} + 1.88$

* assumes a load factor of 0.62 and gas diffusion enrichment except where indicated otherwise.

† does not include any deductions for the sale of unused plutonium. The estimated credit is about $12 per gramme Pu which reduces total non-fuel costs by approximately 0.055 mills per MJ(e).

‡ assumes a load factor of 0.62 and gas centrifuge fuel enrichment.

Note, G = ore grade in % U_3O_8.

Table 8.8 : Average fundamental costs of electricity from a typical PWR system (1965 - 1970 values).

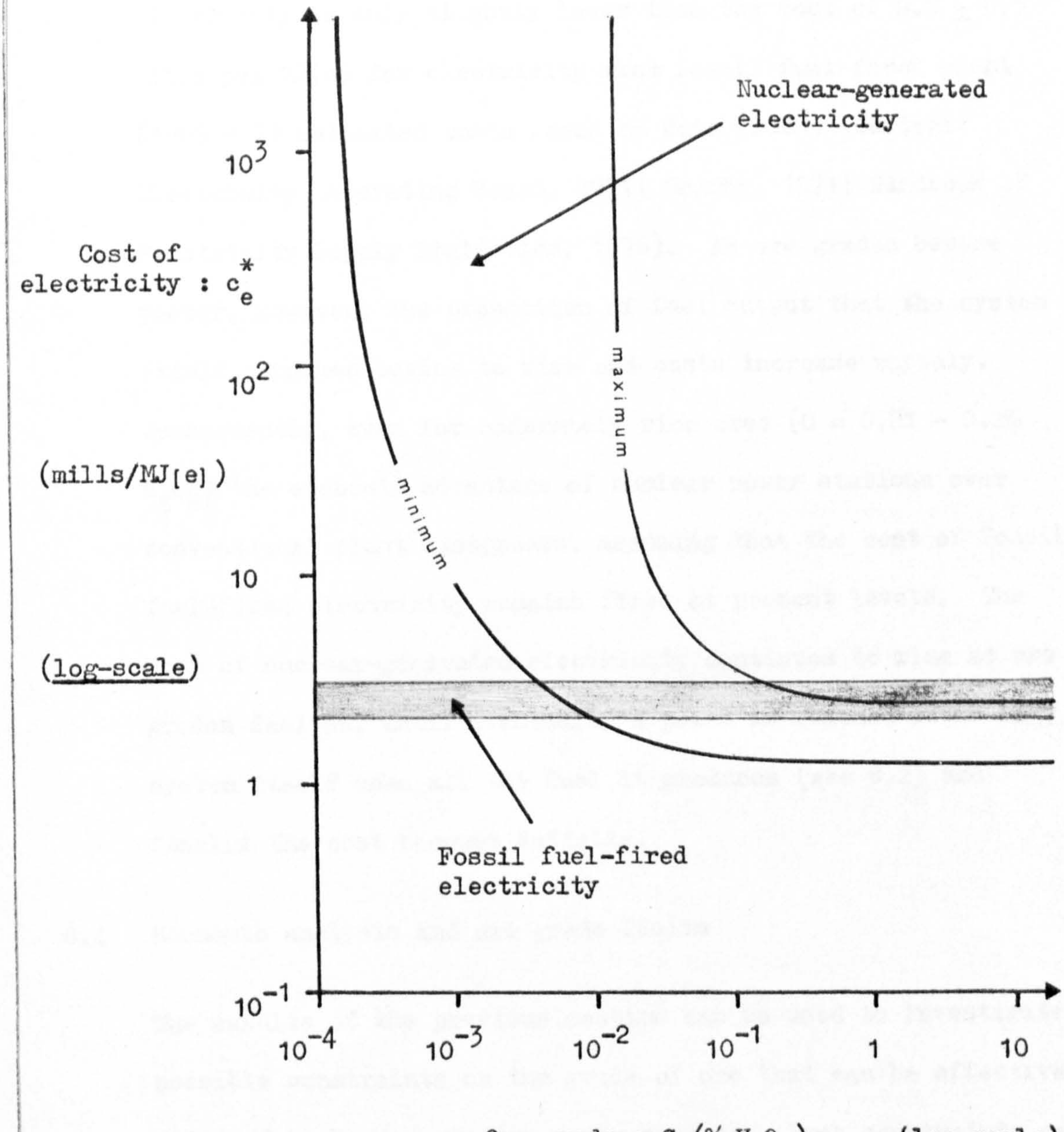
Fuel management policy	Fundamental cost of electricity*: c_e (mills/MJ[e])
Primary fuel cycle†	$\frac{1.83G + 0.0155}{0.919G - 0.0013}$
Secondary fuel cycle with uranium re-cycling†	$\frac{1.84G + 0.0133}{0.920G - 0.0010}$
Secondary fuel cycle with uranium and plutonium re-cycling	$\frac{1.83G + 0.0104}{0.933G - 0.0008}$
Secondary fuel cycle with uranium and plutonium re-cycling‡	$\frac{1.88G + 0.0104}{0.971G - 0.0008}$

* assumes a load factor of 0.62 and gas diffusion enrichment except where indicated otherwise.

† no plutonium credit included.

‡ assumes a load factor of 0.62 and gas centrifuge enrichment.

Figure 8.2 : Effect of ore grade on the fundamental cost of nuclear-generated electricity* (1965 - 1970 values).



* PWR system operating at a load factor of 0.62 and using the primary fuel cycle with gas diffusion enrichment.

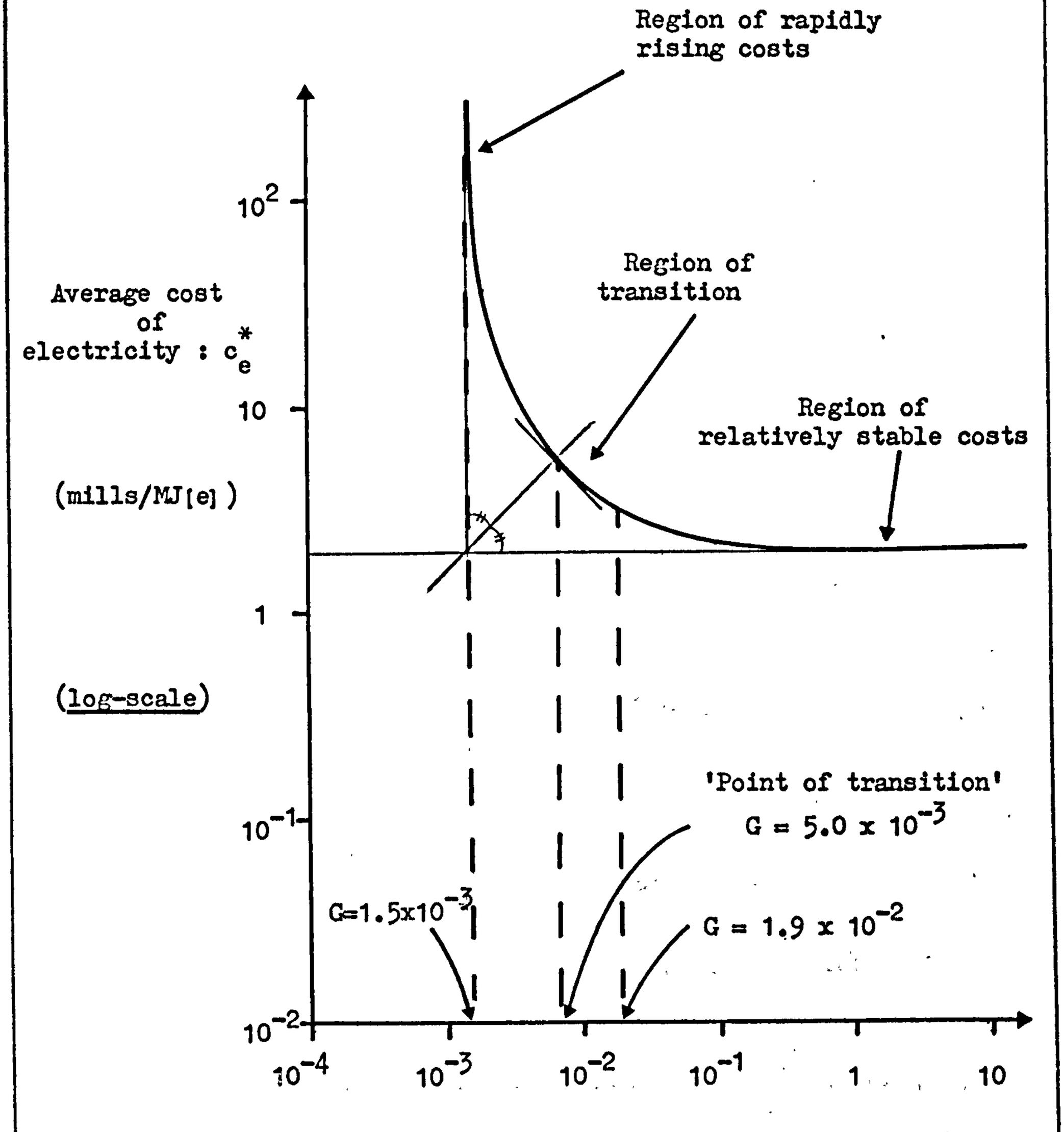
Electricity Supply Statistics, 1976), despite gradually declining average ore grades (eg. in the USA grades fell from a record 0.55% U_3O_8 in 1955 to 0.20% U_3O_8 in 1972 : U.S. Atomic Energy Commission, 1972).

With current ore grades the cost of nuclear-generated electricity is only slightly lower than the cost of 2.5 $\pm$ 0.5 mills per MJ(e) for electricity from fossil fuel-fired plant (1965 - 70 estimated costs based on data from : Central Electricity Generating Board, 1971; Searby, 1971; Handbook of Electricity Supply Statistics, 1976). As ore grades become poorer, however, the proportion of fuel output that the system itself consumes begins to rise and costs increase rapidly. Consequently, even for moderately rich ores (G = 0.01 - 0.2% U_3O_8) the economic advantage of nuclear power stations over conventional plant disappears, assuming that the cost of fossil fuel-fired electricity remains fixed at present levels. The cost of nuclear-generated electricity continues to rise as ore grades decline, until eventually a point is reached where the system itself uses all the fuel it produces (see 8.2) and finally the cost becomes infinite.

8.4 Economic analysis and ore grade limits

The results of the previous section can be used to investigate possible constraints on the grade of ore that can be effectively utilised by burner reactor power systems. Such constraints can be defined by numerous criteria and some of these are demonstrated in figure 8.3 which illustrates the variation of the average cost of electricity with ore grade for a PWR system operating at a load factor of 0.62 and using the primary fuel cycle with gas diffusion enrichment. This figure shows the

Figure 8.3 : Identification of criteria for ore grade limits.



* PWR system operating at a load factor of 0.62 and using the primary fuel cycle with gas diffusion enrichment.

typical transition of costs from a relatively stable region at high grades to a region of rapidly rising costs for grades approaching the ultimate limit (see 8.2).

The ultimate ore grade limit was defined previously as the point where the total fuel output of a system equals its total energy input, that is when n.e.r., R, equals unity. From figure 8.3 it can be seen that this limit also corresponds to the grade at which the cost of electricity is infinite. Obviously this is the ultimate constraint since it means that the system is totally inefficient as a source of fuel. However, the lowest grade of ore which can be used economically by the system is likely to occur well before this ultimate point is reached. For example, the actual economic limit could be specified as the ore grade at which the region of stable costs ends. Unfortunately the cost of electricity is never exactly constant, even for high ore grades, and consequently the stable region must be arbitrarily designated as that portion of the curve for which costs do not vary more than a fixed amount, say 50%. For a typical burner reactor power system this limit occurs between 100 ppm and 800 ppm U_3O_8, on average 200 ppm U_3O_8.

Since this 'stable cost' criterion can only be formulated loosely, a more accurate definition would obviously have more practical value. Figure 8.3 shows that, in addition to a region of fairly constant costs, there is also an area in which costs are highly sensitive to small changes in ore grade. It would probably be unsatisfactory to use such material since slight differences in grade, which are very difficult to detect by current industrial ore control methods, would cause large fluctuations in the cost of electricity. Fortunately a point can be defined which exactly bisects these two regions of the

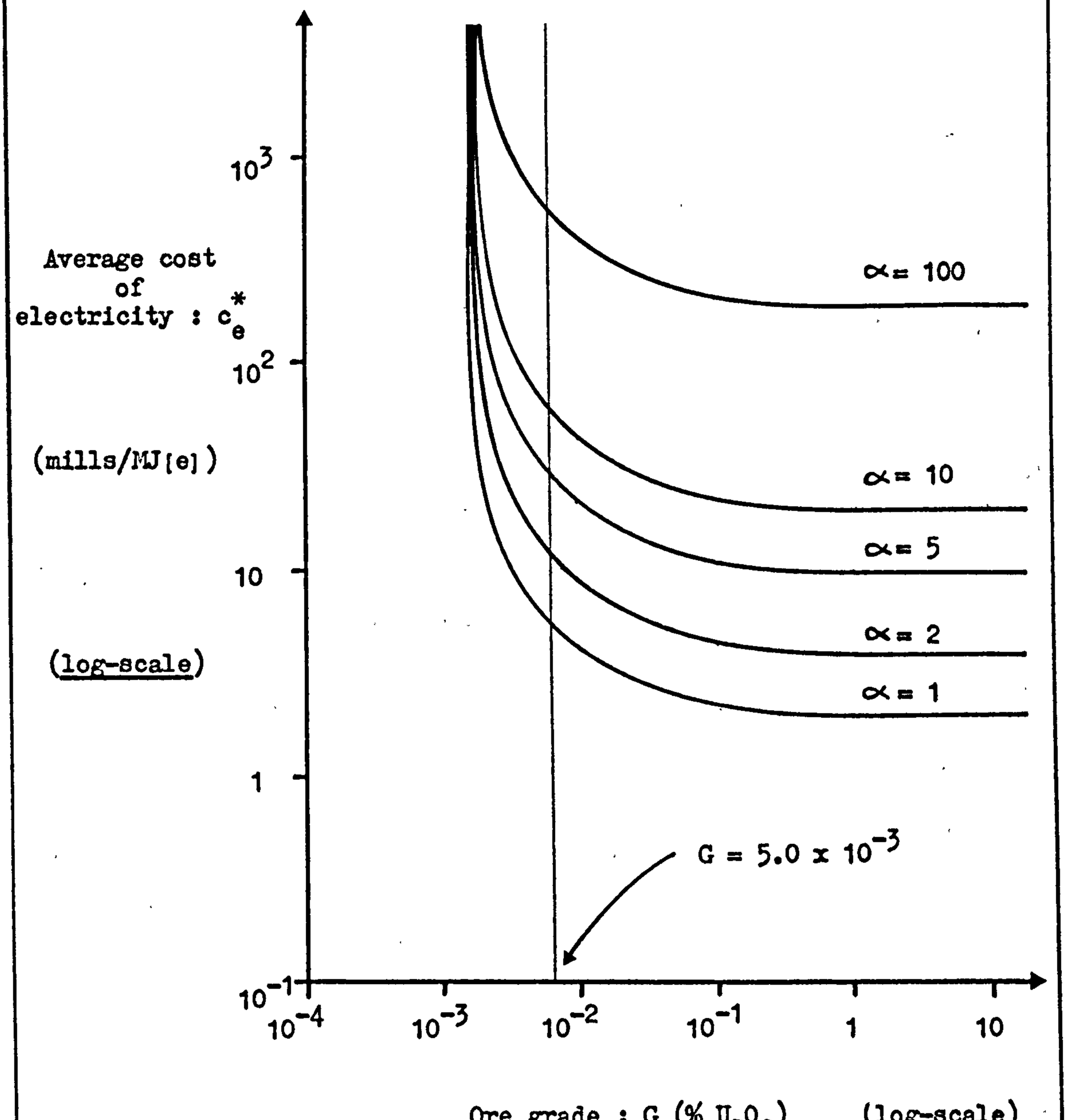
cost curve (see figure 8.3) and this can be used to specify a practical economic ore grade limit. For a typical burner reactor power system this limit ranges from 10 ppm to 400 ppm U_3O_8, with an average value of 50 ppm U_3O_8. This limit is not markedly reduced by using different fuel management policies - the average value for a system with uranium and plutonium mixed oxide re-cycling and gas centrifuge enrichment is 30 ppm U_3O_8.

However, if this ore grade limit is used to assess the proportion of uranium resources that are economically available, then it must be independent of certain basic factors implicitly incorporated into this simple economic model. In particular, the limit should not drastically alter with expected changes in the value of the non-fuel cost, c_o. This cost depends on such items as wages, interest rates, etc., which are unlikely to remain fixed at the current (1965 - 70) levels assumed by the model. For example, the average hourly wage index has already more than trebled between 1968 and 1976 (Annual Abstract of Statistics, 1976).

The effect of the value of non-fuel costs relative to current costs, α, on the ore grade limit is shown in figure 8.4. A wide range of increases in the level of non-fuel costs is investigated and the figure demonstrates that the ore grade limit is distinctly insensitive to such changes. Consequently the results of this analysis were assumed to be fairly applicable to a wide number of situations.

Ore grade limits based on the different criteria examined here are summarised in table 8.9. Estimates of the amount of

Figure 8.4 : Sensitivity of ore grade limit criteria to the relative level of non-fuel costs.



α = factor increase of non-fuel costs in relation to current levels (1965 - 1970).

* PWR system operating at a load factor of 0.62 and using the primary fuel cycle with gas diffusion enrichment.

Table 8.9 : Ore grade limits and available resources.

Criterion	Ore grade limits (ppm U_3O_8)			World uranium resources* (tonnes U_3O_8)	
	minimum	maximum	average	current	extrapolated
Ultimate energy limit; R = 1, $c_e = \infty$:	1.5	150	15	2.2×10^9	3.3×10^9
Arbitrary stable cost; ± 50% :	100	800	200	2.5×10^6	3.2×10^7
Point of transition :	10	400	50	1.0×10^7	4.2×10^7

* based on the average ore grade limit.

uranium available from ores of grades which are higher than the average limit were deduced from data presented in appendix F and results are also included in this table. Two particular types of resource estimate are shown in table 8.9. These are the current world total resource base which consists of uranium from known, inferred and indicated deposits and the extrapolated world total resource base which was evaluated from present data using a very simple resource model (see appendix F). The current estimate indicates the amount of uranium expected to become available during coming years, whilst the extrapolated value represents a rough estimate of the ultimate potential.

Assuming that, for all practical purposes, the actual economic ore grade limit is specifically defined by the point of transition on the electricity cost curve, then table 8.9 shows the average total uranium resources avaliable for burner reactor power systems is between about 1×10^7 tonnes and 4×10^7 tonnes U_3O_8. Minimum and maximum estimates can also be deduced from the extreme limits included in this table. The lower and upper results defined by the 'point of transition', 10 ppm and 400 ppm U_3O_8 respectively, produce corresponding variations in current resources of 4.5×10^6 tonnes to 2.2×10^9 tonnes U_3O_8 and in the extrapolated estimate ranging from 3.2×10^7 tonnes and 3.3×10^9 tonnes U_3O_8. These substantial variations are largely caused by uncertainties in basic ore mining and processing data such as the nature of the ore and deposit characteristics. However, despite potentially large fluctuations in these parameters, information from currently operating mines and mills supports the incorporation of average data into the analysis. Consequently average values of the economic ore grade limit and uranium resources were used in the following interpretation and comparison of results.

9 INTERPRETATION AND COMPARISON OF RESULTS

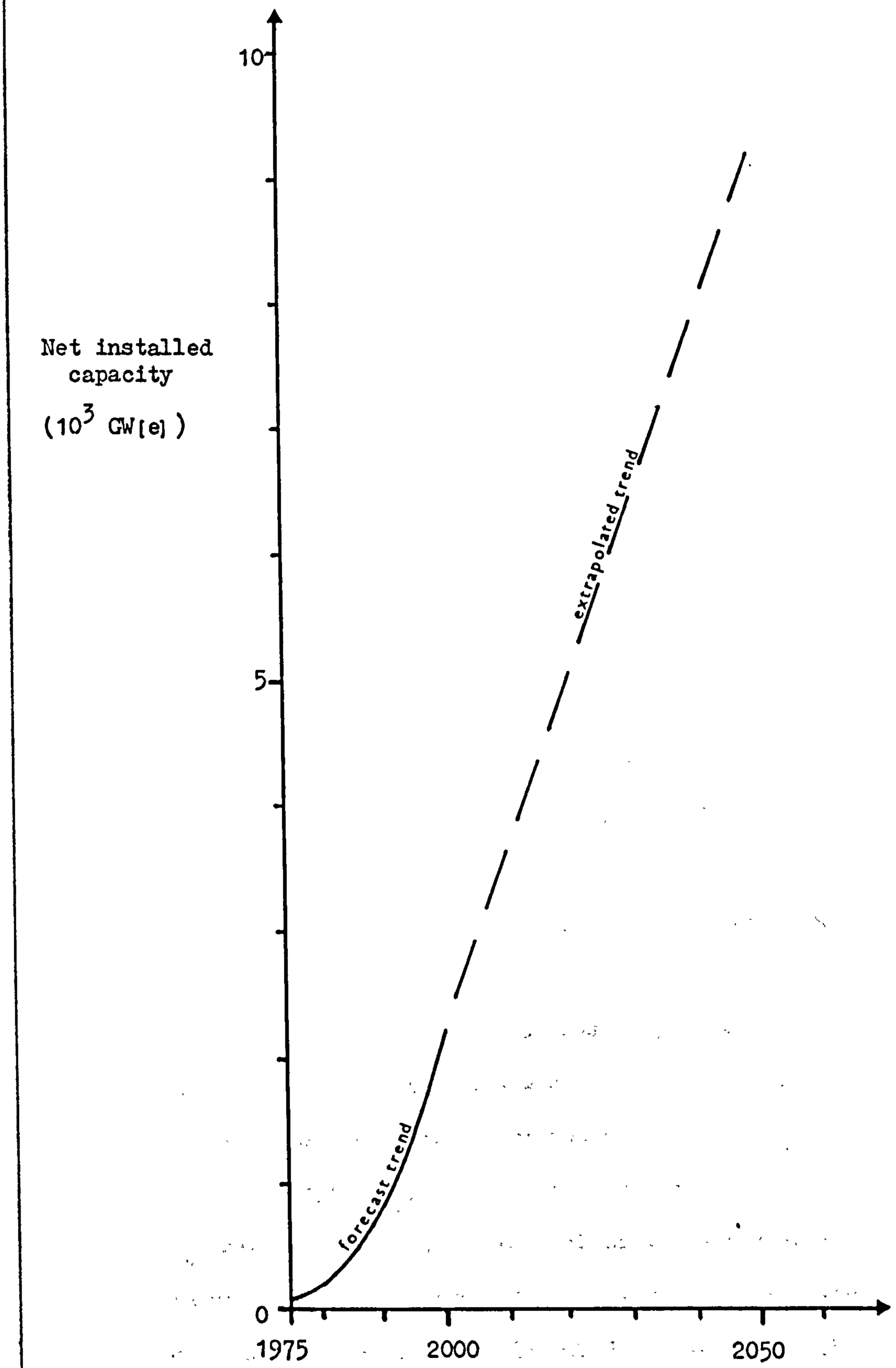
9.1 Introduction

The purpose of this chapter is to investigate the effects of constraints on the quantity of economically recoverable uranium available for use in burner reactors on the development of nuclear power as a significant source of fuel. The resource base estimates obtained from previous analysis are assessed in the light of the expected growth of nuclear capacity and subsequent demand for uranium concentrate. Results are compared with other, more traditional resource assessments and differences are examined in terms of the evaluation of the cost of producing uranium concentrate from naturally-occurring sources.

9.2 Appraisal of uranium resources

Economic limitations on the scope of the uranium resource base demonstrated in the previous chapter obviously impose constraints on the development of the world nuclear power industry in its present form. A typical forecast of future nuclear power plant capacity for the world (excluding the USSR, Eastern Europe and China) is shown in figure 9.1. This is a speculative assessment based primarily on the industry's own expectations of the average estimated growth between 1975 and 2000 (deduced from the latest comprehensive study of international nuclear power development: Nuclear Energy Agency, 1975). The trend beyond 2000 is an extrapolation which simply assumes linear growth with capacity increasing by almost 140 GW(e) each year ($1GW = 10^9$ watts).

Figure 9.1 : Speculative forecast of world* nuclear power capacity growth.

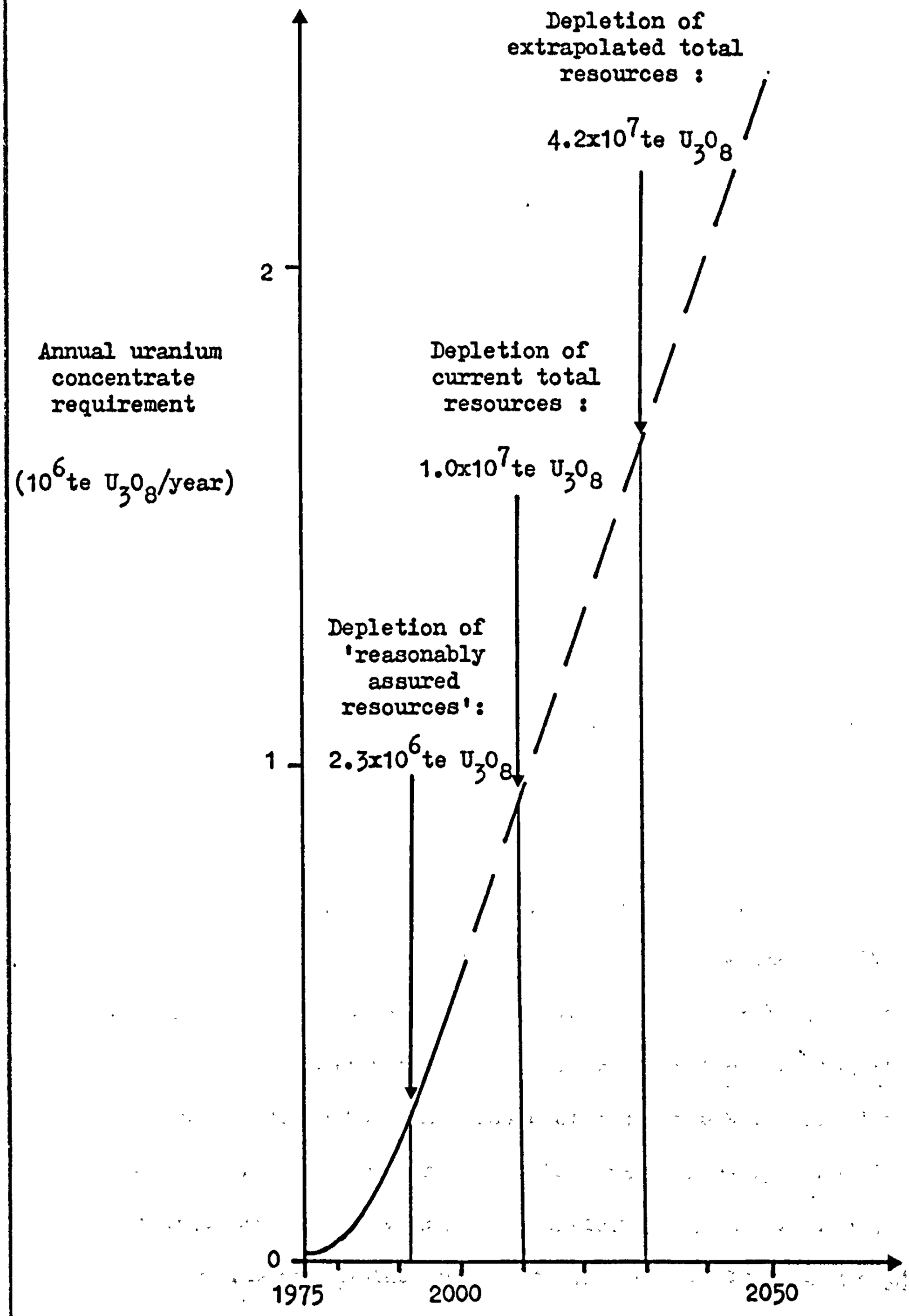


* excluding the USSR, Eastern Europe and China.

The effect of this growth on the amount of uranium actually available for consumption can be investigated by comparing the cumulative use of uranium concentrate with the extent of the resource base. The cumulative amount of uranium concentrate that has been consumed in any given period equals the sum of the annual rates of production in that period. The variation of the annual uranium concentrate output corresponding to the type of growth indicated in figure 9.1 is shown in figure 9.2. This figure was calculated assuming that the majority of reactors in operation are PWR designs which have a core of 101 tonnes of uranium enriched to 2.5% U-235 and which consume 33.8 tonnes of uranium enriched to 3.1% U-235 yearly (see tables 5.16, 6.2 and 6.6). This results in an average annual requirement of 262 tonnes U_3O_8 for each GW(e) of nuclear power capacity installed.

Figure 9.2 compares total requirements with various estimates of available uranium resources. Three particular resource bases are used in this study; 'reasonably assured resources', current total world resources and extrapolated total world resources. Reasonably assured resources correspond to the uranium in currently known ore deposits which could be exploited almost immediately (Nuclear Energy Agency, 1975). Such resources amount to 2.3×10^6 tonnes U_3O_8 (see appendix F). Current total world resources consist of a wide range of ores in known and inferred deposits. For ore grades higher than the economic limit of 50 ppm U_3O_8, these resources contain about 10^7 tonnes U_3O_8 (see appendix F). Since much of these resources occur in surmised deposits, further assessment would be required before such uranium becomes generally available for use. The extrapolated total world resource base which was derived from

Figure 9.2 : Comparison of the world* forecast annual rate of uranium concentrate demand with available resources.



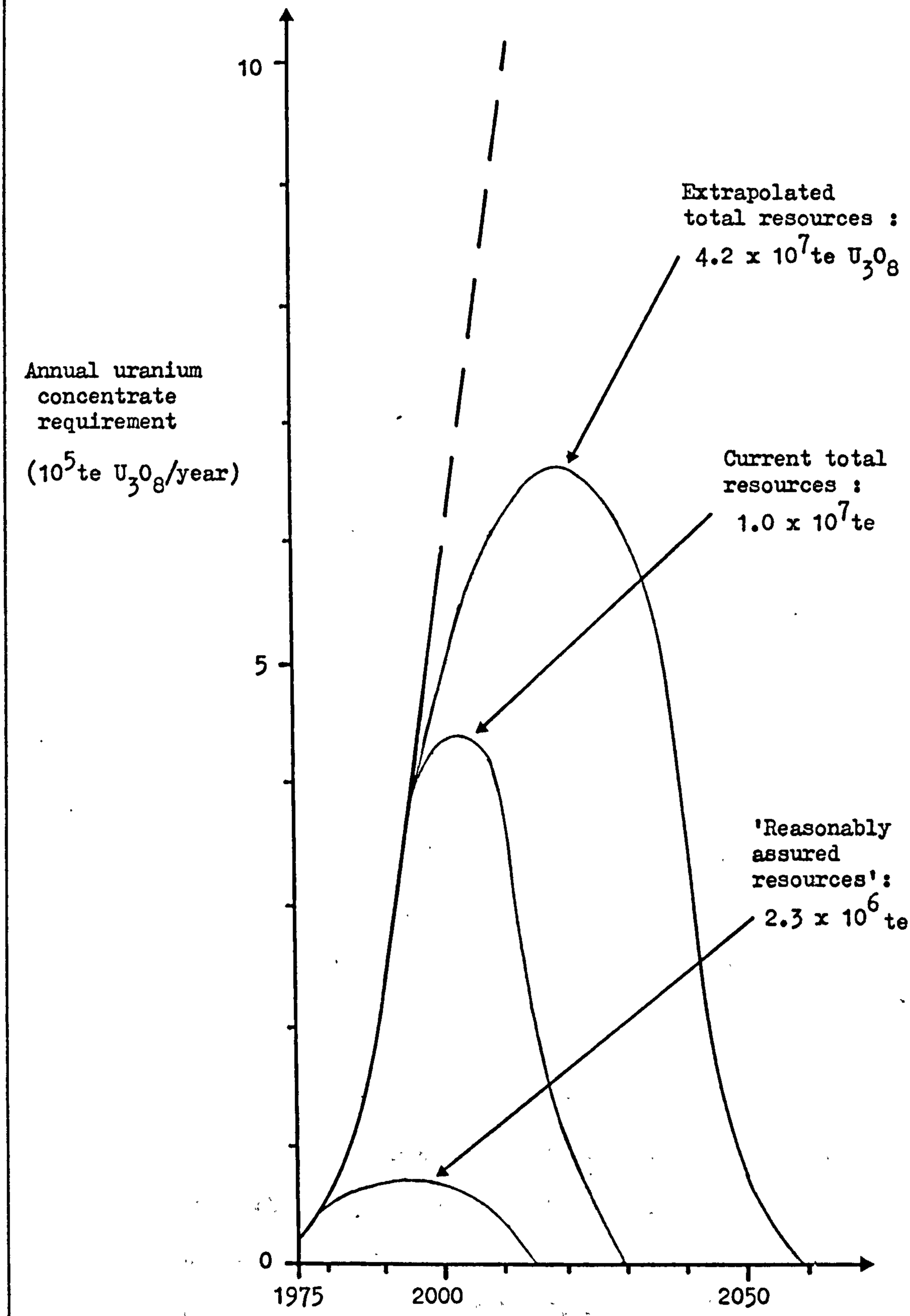
* excluding the USSR, Eastern Europe and China.

a very simple resource model represents an approximate estimate of the ultimate amount of uranium which may be potentially available in the world. Although the majority of these resources are yet to be actually discovered, the total amount involved, for grades higher than 50 ppm U_3O_8, is expected to be about 4.2×10^7 tonnes U_3O_8 (see appendix F).

In figure 9.2 the years during which the demand for uranium concentrate exceeds that available from these particular resource base estimates are clearly indicated. For 'reasonably assured resources' supply fails to meet demand around 1990, for current total resources in 2010 and for extrapolated total resources in about 2030. However, these figures do not accurately portray the likely effects of the depletion of uranium resources, since such shortages of nuclear fuel would suddenly leave a large amount of power plant capacity inactive. A more realistic situation is illustrated in figure 9.3 which shows the consequences of ensuring that all power stations are guaranteed fuel for their entire operational lifetime (for PWR systems this would be about 35 years).

Figure 9.3 presents depletion profiles for the three particular uranium resource bases considered here. A depletion profile is a diagram which demonstrates the effect of the utilisation of materials in relation to the finite scope of their resources. With the specific forecast growth of nuclear power suggested by figure 9.1, these profiles indicate that reasonably assured resources of uranium would be completely exhausted by 2010, although the demand for uranium concentrate would exceed supply at the much earlier date of 1980. Similarly, shortages in the supply of uranium from both the current and extrapolated

Figure 9.3 : World* uranium concentrate demand and the likely depletion of uranium resources.



* excluding the USSR, Eastern Europe and China.

estimates of total world resources would apparently develope in 1995. Sufficient uranium is available from the larger, extrapolated resource base to provide fuel until 2060, but the smaller, current resource base would only last up to about 2030.

In addition to assessing the extent of uranium resources in relation to the demands of forecast growth in burner reactor power capacity, figure 9.3 can also be used to compare uranium with other energy resources. A very basic comparison can be achieved by evaluating the period required to consume the middle 80% portion of the depletion profile which represents the majority of available resources (cf. Hubbert, 1969). Figure 9.3 indicates that this period would be between 25 and 45 years for uranium resources used in burner reactors. By contrast the equivalent period for world crude oil resources has been estimated as about 60 years, although a substantial proportion of these resources have already been consumed (Hubbert, 1969). Similarly, the period needed to exploit all petroleum-based fuels is about 100 years and that for world coal resources is between 300 and 400 years, or only 100 to 200 years if coal is used as the main energy source (Hubbert, 1969).*

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* Note that all resources would be able to support similar peak levels of equivalent electrical capacity;

Uranium (in burners) = 2×10^3 to 5×10^3 GW(e)

Oil = 1×10^3 to 2×10^3 GW(e)

Coal = 4×10^3 to 5×10^3 GW(e)

9.3 Comparison of resource estimates and evaluation of ore costs

The foregoing assessment implies that the currently and potentially available amount of economically recoverable uranium resources for use in burner reactor power systems are rather limited as regards the lifespan of other resources and the type of world growth in nuclear power capacity expected by international planning organisations and the nuclear power industry. This conclusion appears to contradict the findings of certain other investigations.

For example, this study indicates that the lowest grade of ore that can be used economically by burner reactors is, on average, 50 ppm U_3O_8 and that the ultimate physical limit is typically 15 ppm U_3O_8. Some researchers have suggested that the economic and physical bounds are much lower - in particular, Brown and Silver (1955) conclude, from order-of-magnitude calculations, '..that most of the granitic rocks in the earth's crust [on average containing 4 ppm U_3O_8] are at mankind's disposal and can be processed for a net energy profit'. The opinions of other authorities, however, tend to support the conclusions of the present analysis -- Bowie (1974); 'the cut-off grade [ore grade limit] that may eventually be reached is difficult to estimate but it could easily be of the order of 50 parts/10^6 metal [60 ppm U_3O_8]'.

Estimates of the resource base for burner reactors obtained by some investigators disagree with the total of 10^7 to 4×10^7 tonnes U_3O_8 deduced here. Lewis (1964) claims that 'since the total abundance of uranium in the land mass of the earth's crust is estimated at 10^{14} tonnes and at least 2×10^{12} tonnes within a mile of the surface, ... a very small fraction of the

resources would be needed for power in hundreds of centuries from thermal fission [burner] reactors...'. Additionally, Brown and Silver (1955) assert '...that the reserves of uranium (and thorium) available to man can be considered for all practical purposes as infinite'.

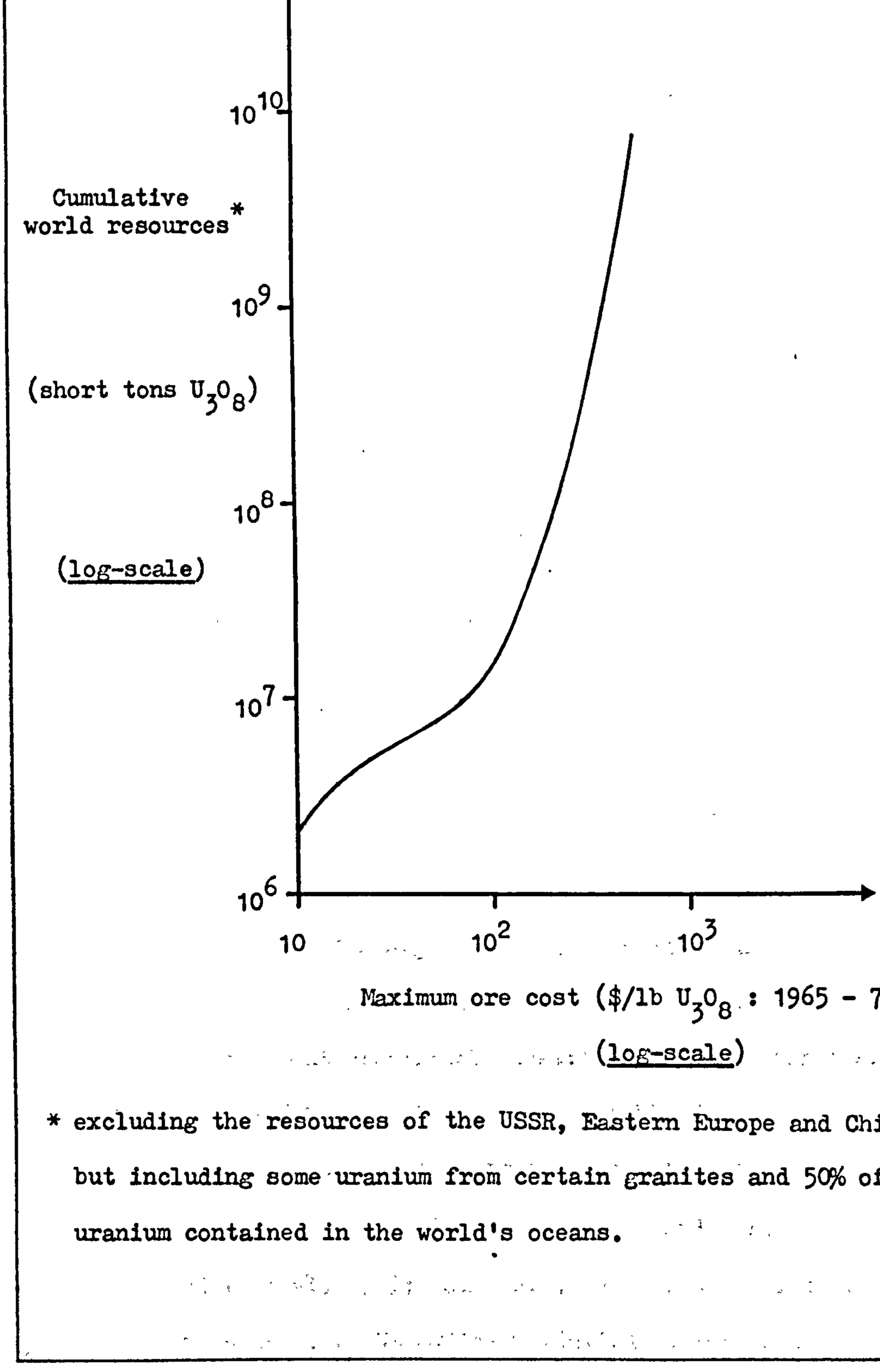
In similar vein the lifetime of uranium resources in relation to burner reactor fuel use is expected to be longer than the full period of between 55 and 85 years estimated in this study. Hurwitz (1974) reports that 'these uranium supplies appear adequate to accommodate the [nuclear power] industry without breeders at least into the first part of the twenty-first century' (cf. figure 9.3). Park (1975) affirms that 'if the world's uranium is to be used in fission reactors like those now in operation and under construction [mainly burner reactors] , the supply of radioactive material, uranium-235, will be exhausted at about the same time as the world's coal supplies [which, according to Hubbert (1969), would be between 100 and 400 years]'.

These significant disparities are probably caused by differences in the general conception of the role of ore costs in nuclear power economics. It is an apparently fundamental tenet of some authorities that every type of resource can be assigned a finite cost and that, provided the consumer is willing to pay the price, even common rocks and seawater which contain minute traces of uranium are perfectly practical sources of nuclear fuel for burner reactors. This belief is supported, to some extent, by traditional resource assessments which indicate that, as the cost of producing uranium concentrate from ore rises, the total, or cumulative, amount of uranium available for use increases without check.

A commonly accepted resource evaluation (Hunt, 1974; Vaughan, 1975) is presented in figure 9.4 which demonstrates the traditionally estimated variation of cumulative uranium resources with the maximum cost of extracting uranium concentrate from naturally-occurring sources such as ores, crustal rocks and seawater. For convenience, uranium resources are measured in this figure in the widely-used units of short tons U_3O_8 (1 short ton = 2000 lb = 0.907 tonnes) and costs are written in terms of US dollars (1965 - 70 values) per pound mass U_3O_8 ($/lb U_3O_8 = $2.6/kg U).

Although figure 9.4 is based on very thorough analyses of the cost of producing uranium from numerous sources, it implies that economically recoverable uranium resources exceed 10^{10} tonnes U_3O_8. This obviously contradicts the results of this study which suggests that the physical limit to resources is about 10^9 tonnes U_3O_8 and the economic limit is in the region of 10^7 tonnes U_3O_8. The reason for this contrast is caused by the fact that the economic studies which form the basis of figure 9.4 indicate the present cost of exploiting various sources of uranium. Consequently fixed fuel costs evaluated with current prices are incorporated into the results of these economic studies. However, as indicated in chapter 8, the cost of producing uranium concentrate depends on the price of fuels and the cost of electricity generated from nuclear sources is also a function of the price of uranium ore. This two-way relationship suggests that, as ore grades decline, not only does the amount of fuel used in exploration, mining and processing rise but also the price of the fuel, if it is nuclear-generated electricity, increases. By ignoring this important feedback mechanism the effect of grade on ore costs is increasingly underestimated as ore grades fall.

Figure 9.4 : Traditional variation of cumulative world uranium resources* with ore costs.



* excluding the resources of the USSR, Eastern Europe and China, but including some uranium from certain granites and 50% of the uranium contained in the world's oceans.

The actual variation of the fundamental ore cost with grade can be deduced by using the results of energy analysis. Assuming that the only fuel consumed by all operations involved in the production of uranium concentrate from ore is electricity and that all this fuel is supplied by burner reactor power stations, then the ore cost can be represented by the following expression;

c_c = cost of producing uranium concentrate

= ore cost

$$= (E_u \times p_e) + c_n$$

where,

E_u = n.e.r. of uranium concentrate (see 5.2.4)

p_e = price of nuclear-generated electricity

c_n = non-fuel costs of uranium concentrate production

If $p_e = c_e$ = cost of nuclear-generated electricity then,

$$c_c = (E_u \times c_e) + c_n$$

This fundamental cost incorporates the effect of self-induced increases in the price of fuels, but does not include rises in the price of other inputs. Such items are calculated at current (1965 - 70) levels.

The amount of energy required to produce uranium concentrate, E_u, was evaluated earlier (see 5.2.4.) and values incorporating the relevant electrical substitution factor for thermal energy, j, (see 8.2) are illustrated in table 9.1. Estimates of the non-fuel cost of concentrate production, c_n, were obtained from economic data (eg. Engineering and Mining Journal, 1961; Bader et al, 1969; Youngberg, 1973) and results are summarised in table 9.2. Using the variation of the cost of

Table 9.1 : Energy requirement of producing uranium concentrate from ores.*

Value	Energy requirement : E_u (MJ[e]/lb U_3O_8)
Minimum :	$\frac{6.9}{G} + 0.2$
Maximum :	$\frac{605}{G} + 944$
Average :	$\frac{68.5}{G} + 13.5$

* including exploration, mining and ore processing, and assuming that electricity is the only fuel used in these operations.

Note, G = ore grade in % U_3O_8.

Table 9.2 : Non-fuel costs of producing uranium concentrate from ores.*

Value	Non-fuel cost : c_n ($/lb U_3O_8 : 1965 - 70)
Minimum :	$\frac{0.43}{G} + 0.50$
Maximum :	$\frac{3.20}{G} + 1.00$
Average :	$\frac{0.75}{G} + 0.75$

* including exploration, mining and ore processing.

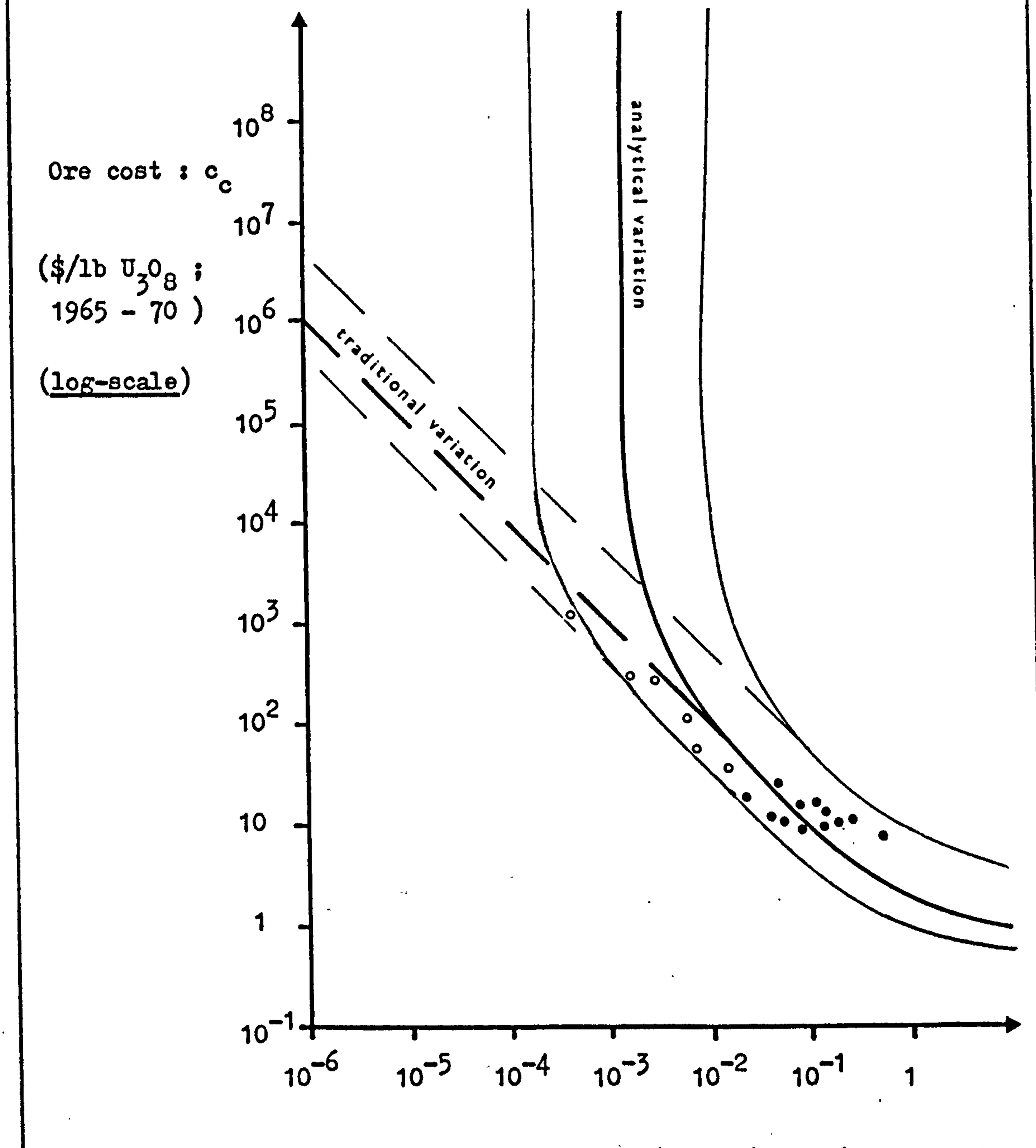
Note, G = ore grade in % U_3O_8.

nuclear-generated electricity, c_e, with ore grade, G, shown in table 8.8 and figure 8.2, the ore cost, c_c, was deduced. Results are illustrated in figure 9.5 which also demonstrates the traditional variation of cost with grade assuming a current (1965 - 70) fixed price for electricity of 2.5 $\pm$ 0.5 mills per MJ(e). The data points included in figure 9.5 refer to present estimates of the cost of concentrate from currently operating plants and new proposed schemes (eg. Brown and Silver, 1955; Engineering and Mining Journal, 1961; Pinkney and Westwood, 1961; Cambel, 1965; Hurst et al, 1966; Bieniewski et al, 1971; Andersson and Olsson, 1975).

Figure 9.5 indicates that, unlike traditional assessments based on present prices, ore costs rise steeply with declining grades to reach an ultimate limit between 1.5 ppm and 150 ppm U_3O_8, with an average of 15 ppm U_3O_8. Hence there are real constraints on the quality of ore that can be used in burner reactors. Practical ore grade limits subsequently reduce the quantity of uranium economically available from resources and figure 9.6 demonstrates this re-appraisal with the variation of cumulative resources with ore costs based in the results of energy analysis.

Differences between traditional and re-appraised estimates of cumulative resources may be examined by comparing figure 9.6 with figure 9.4. Disparities are partly a result of the re-evaluation of ore costs. Additionally, uranium resources contained in seawater at a concentration of 3.3 ppb U are not included in figure 9.6. A similar analysis to that applied to ores may also be used to assess the cost of producing uranium from seawater by methods which are currently under development

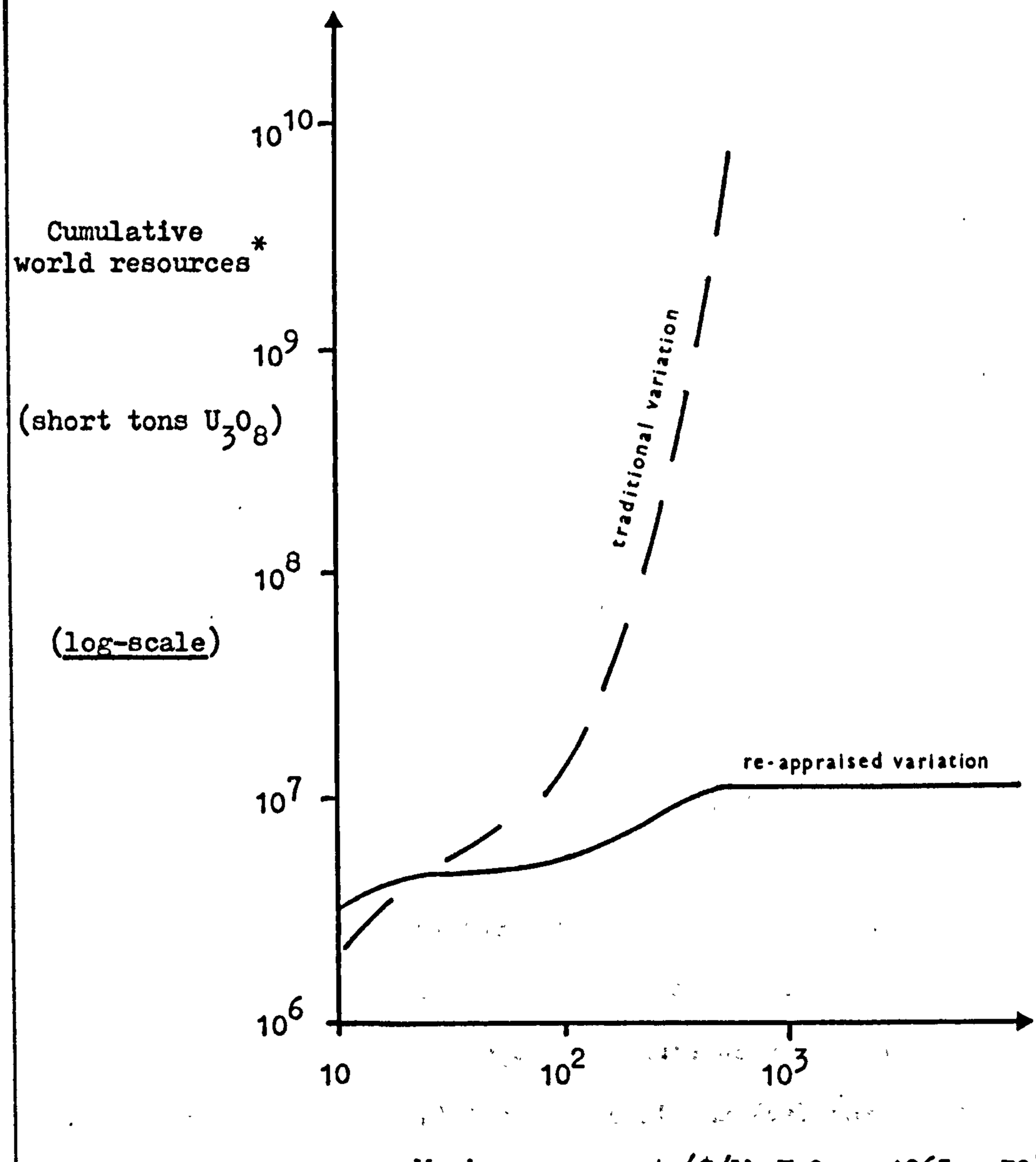
Figure 9.5 : Variation of the cost of producing uranium concentrate from ore with ore grade.



Data points ; • = actual operations

o = proposed schemes

Figure 9.6 : Re-appraised variation of current cumulative world uranium resources* with ore costs.



* excluding the resources of the USSR, Eastern Europe and China.

(see appendix E). Although initial studies of seawater treatment processes obtained order-of-magnitude costs between \$10 and \$100 per pound U_3O_8 (Davies et al, 1964; Keen, 1968; Bieniewski et al, 1971), a more detailed, though traditional economic assessment assuming fixed fuel prices of 3.6 mills per MJ(e) indicated a minimum cost of \$260 per pound U_3O_8 (Harrington et al, 1974). By contrast, a cost of \$500 per pound U_3O_8 for extraction techniques relying on tidal flow schemes was deduced from energy analysis.

The use of such uranium in burner reactors would result in electricity costs of about 8.4 mills per MJ(e) (1965 - 70 values). Figure 8.3 demonstrates that this is higher than the cost of electricity (7.5 mills per MJ[e]) from ores of grade less than the practical economic limit (50 ppm U_3O_8). Consequently, since tidal flow extraction schemes can only achieve relatively poor rates of return on capital investment and as production is limited by the number of suitable sites (see appendix E), it seems unlikely that much uranium will be obtained from the world's oceans by such methods. Hence these resources are excluded from figure 9.6. It should also be noted that low capital cost, compact seawater processing schemes which achieve high production rates by pumping require more electrical energy to produce uranium than a burner reactor using such material can generate as fuel (see appendix E). Therefore it is unlikely that such pumping schemes will make the vast resources of the oceans available for use in burner reactor power systems.

10 CONCLUSIONS

10.1 Summary of results

This study has demonstrated the general basic principles and type of information used to evaluate the amount of energy required in the production of electricity from uranium by nuclear fission in burner reactor power systems. The energy analysis indicates that such systems currently consume substantially less energy in the form of fossil fuels than systems based on conventional coal-, gas-and oil-fired power plant. Additionally, the results show that burner reactor power systems can be regarded as net producers of fuel with currently mined ores, since such systems also consume much less energy than they generate.

However, this investigation illustrates that the net energy requirement of nuclear-generated electricity, or energy input per unit fuel output, depends on numerous factors such as the quality, or grade, of the ore, reactor design, fuel cycle technology, fuel management policies and the way in which power stations are operated. The quality of ore, represented by the percentage uranium content, or ore grade, is the most fundamental variable since a decline in this particular parameter can cause the amount of fuel used by the system to rise above that consumed by fossil fuel-fired systems and even exceed the electrical energy output of the nuclear system itself. The role of reactor type, enrichment technology, fuel re-cycling and power station load factor in determining the net energy requirement of nuclear-generated electricity is shown to be important but secondary in comparison with the effect of the uranium ore grade.

Evaluation of the significance of certain improvements and changes in the the nuclear power industry for fuel conservation is an obvious practical application of the results of energy analysis. Attractive potential fuel savings can be achieved by many measures which include improving the use of fuel in ore mining and processing, installing new, more efficient isotope enrichment plants and re-cycling unused uranium and plutonium from spent nuclear material. The results indicate that the use of the gas centrifuge enrichment technique instead of the widespread gas diffusion process would introduce the largest savings at present, although in the future fuel conservation in ore mining and processing could produce even greater reductions. Further examination, however, suggests that practical considerations such as the lack of incentives, long construction lead times and high capital expenditure may delay the implimentation of fuel conservation schemes.

Investigation of the constraints on the sources of uranium that can be used effectively and profitably by burner reactor power systems is probably the most important feature of this study. The results of energy analysis demonstrate that there is an ultimate limit to the grade of ore which can be utilised by such systems. A practical economic boundary is indicated by the variation, with ore grade, of the cost of nuclear-generated electricity based on a very simple economic model which incorporates energy analysis results.
Limitations on the type of material that can be consumed by burner reactors impose constraints on the quantity of uranium which is physically and economically recoverable from resources. This suggests that the uranium resource base is not only

finite but also relatively small. Hence serious shortages may well develope before the turn of the century if the current forecasts of burner reactor power capacity growth materialise. Furthermore, the results demonstrate that the lifespan of uranium resources used in burner reactors alone will be considerably less than the predicted lifespan of fossil fuels.

Although the resource assessment of this study contradicts the findings of certain other previous investigations, results are used to explain these disparities. In particular ore costs, or the cost of producing uranium concentrate from natural sources, deduced by traditional methods which incorporate fixed, present-day fuel prices are compared with ore costs evaluated from the results of energy analysis. Such costs take into account the fact that the price of ore can determine the cost of nuclear-generated electricity and that ore costs are a function of the price of fuels, including electricity. Examination shows that traditional assessments inadvertently over-estimate the quality and quantity of uranium that can be used to provide fuel from burner reactors. Such results invite the conclusion that all attempts to evaluate the realistic extent of any energy resource should recognise that the price of fuel obtained from such resources is likely to determine the cost at which they can be exploited. This study demonstrates that energy analysis is a fundamental means of exploring this vital, self-reinforcing link.

10.2 A brief summary of similar work

Some aspects of the present study were originally discussed in a joint research report entitled "Energy inputs and outputs for nuclear power stations" (Chapman and Mortimer, 1974), which was first published in September 1974 and then revised in December 1974. The principal aim of this report was 'to describe a method of analysing nuclear power systems so that some of the physical consequences of decisions can be understood'. This analysis consisted of evaluating the amount of energy consumed by individual power stations and of investigating the effect on total fuel supply of building a series of power stations, that is, studying the dynamic energy analysis of power programmes.

The main conclusions of the report were that, at the moment, burner reactor power systems use less energy than conventional fossil fuel-fired power systems, that the amount of uranium available from resources for such nuclear systems ultimately depends on the total fuel efficiency which is a function of uranium ore grade, and, that the rate of growth of power station construction programmes can significantly alter the net power production or consumption of the entire system. The report and its conclusions were later summarised in "The ins and outs of nuclear power" (Chapman, 1974) and "Energy analysis of nuclear power stations" (Chapman, 1975).

Following this original report, its calculations and subsequent results were checked, using the same data, in "Dynamic energy analysis and nuclear power" (Price, 1974) and "Nuclear aspects of energy accounting" (Hill and Walford, 1975a). Additionally the analysis has been re-worked by other researchers using

different data. Similar results to those of the original study were obtained by Creagan (1974) in the Westinghouse Electric Corporation report "Net output of energy from nuclear sources", by Charpentier (1975) in "Toward a better understanding of energy consumption" and by Stöhl in a verbal report to an international workshop on energy analysis and economics (IFIAS, 1975). The only significant differences in results appear to have been discovered by Davies and the staff of the Bechtel Corporation whose initial claims, reported in "A nuclear plant pays back its energy investment in 2 - 3 months" (Nucleonics Weekly, 1975), were based on, as yet, unpublished information.

Other reports which have attempted detailed analysis include "Net energy from nuclear reactors" (von Hippel, Fels and Krugmann, 1975), "Net energy from nuclear power" (Institute for Energy Analysis, 1975), "The energy budget of nuclear power" (Sørensen and Linderstrøm-Lang, 1975), "Energie-analyse van de totale kernenergie cyclus gebaseerd op licht water reactoren" (Kistemaker, 1975) and "Der Energieaufwand für den Bau und Betrieb von Kernkraftwerken" (Kolb, Niehaus, St. Rath-Nagel and Voss, 1975).

A number of studies have investigated the effect on fuel supply of various general and specific nuclear power programmes. Extension of the original analysis was provided by Price (1974) and a re-appraisal of conclusions, in the light of realistic rates of growth, has been presented by Leach (1974) in "Nuclear energy balances in a world with ceilings". General principles and results were re-examined in relation to fossil fuel savings by Wright and Syrett (1975) in "Energy analysis of nuclear power". The consequences of building a power system based on different types of plant have been

studied by Hill and Walford (1975b) in the report "Energy analysis of a power generating system" which was based on data presented in "Energy cost of inputs to nuclear power" (Walford, Atherton and Hill, 1976). The fuel supply aspects of actual national power programmes have also been assessed by various reports which include "An input-output study of projections for nuclear power growth" (Merriam, 1974), "Energy requirements for a nuclear power programme" (Crouch and Eden, 1975) and "Sixth Report: Nuclear power and the environment" (Royal Commission on Environmental Pollution, 1976).

A few other studies have examined other features of nuclear power systems with energy analysis. Taylor and Walford (1974) in the Programmes Analysis Unit report "Uranium from seawater: an energy cost study" deduced the energy requirement of producing uranium concentrate from seawater and assessed the implications of using such material in burner reactors. Both tidal flow and pumped schemes involving low rates of return on capital were considered and subsequent results agree with the estimates of this study (see appendix E). The effect on uranium resource availability of the ore grade constraints evaluated by energy analysis has been indicated by Chapman (1976a) in "Are the energy resources guaranteed by nuclear power?". In addition, the basic principles of estimating the effect of uranium ore grade on the costs of nuclear power have been outlined in "The all-electric dream" (Chapman, 1976b).

10.3 Suggestions for further work

Energy analysis has been used in this study to assess the effect of fuel conservation schemes in burner reactor power systems. The role of fundamental factors which impose physical and economic restrictions on the type and amount of uranium available for such systems has also been investigated. From the results obtained it appears that energy analysis is a particularly useful technique for studying various aspects of the production of fuel from energy resources and obviously such methods can be applied to other fuel supply systems as well as the current nuclear power industry. Consequently this sort of analysis should aid energy resource assessment, help in the appraisal of new resource extraction processes and provide important information for the formulation of fuel policy.

Specific results of this study indicate that fuel resources for burner reactors are seriously limited in relation to the expected growth in uranium demand and current estimates of uranium resources. This implies that more efficient methods of using uranium other than burner reactors, which, at best, are only capable of consuming 0.71% of the total mass, are required if substantial amounts of electricity are to be generated by nuclear fission up to, and during, the next century. Better uranium utilisation can be achieved by new reactors such as the breeder and converter designs. In principle the breeder reactor uses uranium much more effectively since it can create fissile plutonium from non-fissile U-238, which accounts for 99.28% of the total mass of natural uranium. In contrast, the converter reactor conserves uranium by producing fissile U-233 from non-fissile thorium.

It has been suggested that both these designs will eventually provide cheap and abundant energy in the future. Although no analysis has been performed on the thorium-fuelled converter reactor, some initial results have been obtained for the uranium-based fast breeder reactor. Basic estimates indicate that the ultimate physical ore grade limit for the type of breeder likely to be used in future power systems, that is, a reactor with a mixed natural uranium and plutonium dioxide core surrounded by a natural uranium 'blanket', is about 300 ppb U_3O_8. The economic ore grade limit would be approximately 1 ppm U_3O_8 which implies a resource base of roughly 3×10^{11} tonnes U_3O_8. Such a vast amount of uranium could support the expected growth in world fuel demand for many centuries, assuming technical and environmental aspects of the breeder fuel cycle could be solved. However, there are a number of important restrictions on the use of breeder reactors, such as projected costs, fuel availability and system growth rates, which further energy analysis can identify and quantify.

Obviously more detailed study of uranium-fuelled breeder and thorium-fuelled converter reactor power systems is required to examine the physical implications of future development of the nuclear power industry. However, the fundamental reasoning behind energy analysis is not simply confined to the evaluation of fuel use and its consequences. Although there are few raw materials that are as ubiquitously used and as irreplaceable as fuels, the generalised principles of energy analysis could also be applied to other items consumed by industrialised society. For example, the use of copper in machinery, plant and equipment which extract, refine and manufacture this important metal from natural sources could be investigated to deduce

fundamental constraints on economically recoverable copper resources. The resources of various inputs could be assessed independently, or even jointly, in this manner and eventually it may be possible to describe certain aspects of the operation of economic systems entirely in physical terms. The value of such detailed information for understanding the relationship between industrial activity and resources could prove to be considerable.

Appendix A : Drilling and blasting

Rock boring, or drilling, and the fragmentation of rock with explosives, or blasting, are fundamental operations in conventional mining, tunnelling and quarrying. Drilling can also be used separately in exploration to sample the ground for ore deposits. Since numerous techniques can be used in both exploration and rock breaking activities, the energy requirements of each can vary widely. Drill speed, hole diameter and length, bit configuration, equipment design and mechanical action can all affect the direct and indirect energy inputs to drilling. Energy inputs to blasting can be influenced by the type of explosive and blast pattern used. Moreover, the energy requirements of both operations also depend on factors such as ground hardness, rock composition and fragmentation characteristics.

Energy requirements of exploratory drilling and mine drilling and blasting were obtained from the widespread information available from research literature, industrial reports, mining articles, etc. Some direct energy requirement results for drilling, which were estimated from the fuel consumption data of different machines operating under various conditions, are illustrated in figure A.1. An extensive range of data was used to formulate this figure (eg. Dare, 1960b; Misra, 1960; Engineering and Mining Journal, 1966; Sinclair, 1969; Chitwood, 1971; Mining Magazine, 1971a; Levettus and Cagnionde, 1974; Mining Magazine, 1975c, 1975d, 1976).

Indirect energy inputs due to the use of drill bits, spare parts and other items were deduced by combining basic operating data (eg. Dewey, 1961; Sinclair, 1969; Chitwood, 1971; Levettus and Cagnionde, 1974; Mining Magazine, 1974, 1975c, 1975e) with specific results from the data base (see chapter 3) which are summarised in table A.1.

A typical range of gross energy requirements for most kinds of rock drilling was evaluated with this information and results for exploratory drilling in particular are shown in table A.2.

The contribution of drilling to the gross energy requirement of breaking a unit mass of rock was deduced by multiplying drilling g.e.r. results, given in terms of energy per unit length of hole drilled, by estimates of the drill penetration rate, or total combined length of holes drilled prior to blasting each unit of rock. Typical rates for various mines with widely differing ground conditions were obtained from mining literature (eg. Barrett et al, 1958; Dare, 1959a, 1959b, 1960a, 1960b; Dare and Lindstrom, 1960; McCutcheon and Futterer, 1960; Pearse, 1961; Everett, 1962; Olsson, 1964; Mining Magazine, 1975b, 1975f) and values are illustrated in figure A.2.

The effect of blasting operations on the total energy input to rock breaking was found by analysing the production and use of explosives. The energy requirements of explosives and ancillary items were obtained from the data base and results are shown in table A.3. The variation of explosive consumption with rock fragmentation size presented in figure A.3 was deduced from numerous

sources of data (eg. Barrett et al, 1958; Dare, 1959a, 1959b, 1960a, 1960b; Dare and Lindstrom, 1960; McCutcheon and Futterer, 1960; Pearse, 1961; Everett, 1962; Olsson, 1964; Hurst, Crouse, Brown and Ross, 1966; Mining Magazine, 1975a, 1975b, 1975f).

The gross energy requirements of rock breaking under various conditions were calculated from the above results and the subsequent range of values, which reflect differences in many fundamental factors, is illustrated in table A.4 and A.5. Table A.4 shows the minimum and maximum estimates for underground small-bore percussion drilling and blasting. Table A.5 indicates a similar variation for surface medium-bore drilling and blasting.

Figure A.1 : Direct energy requirement of rock drilling

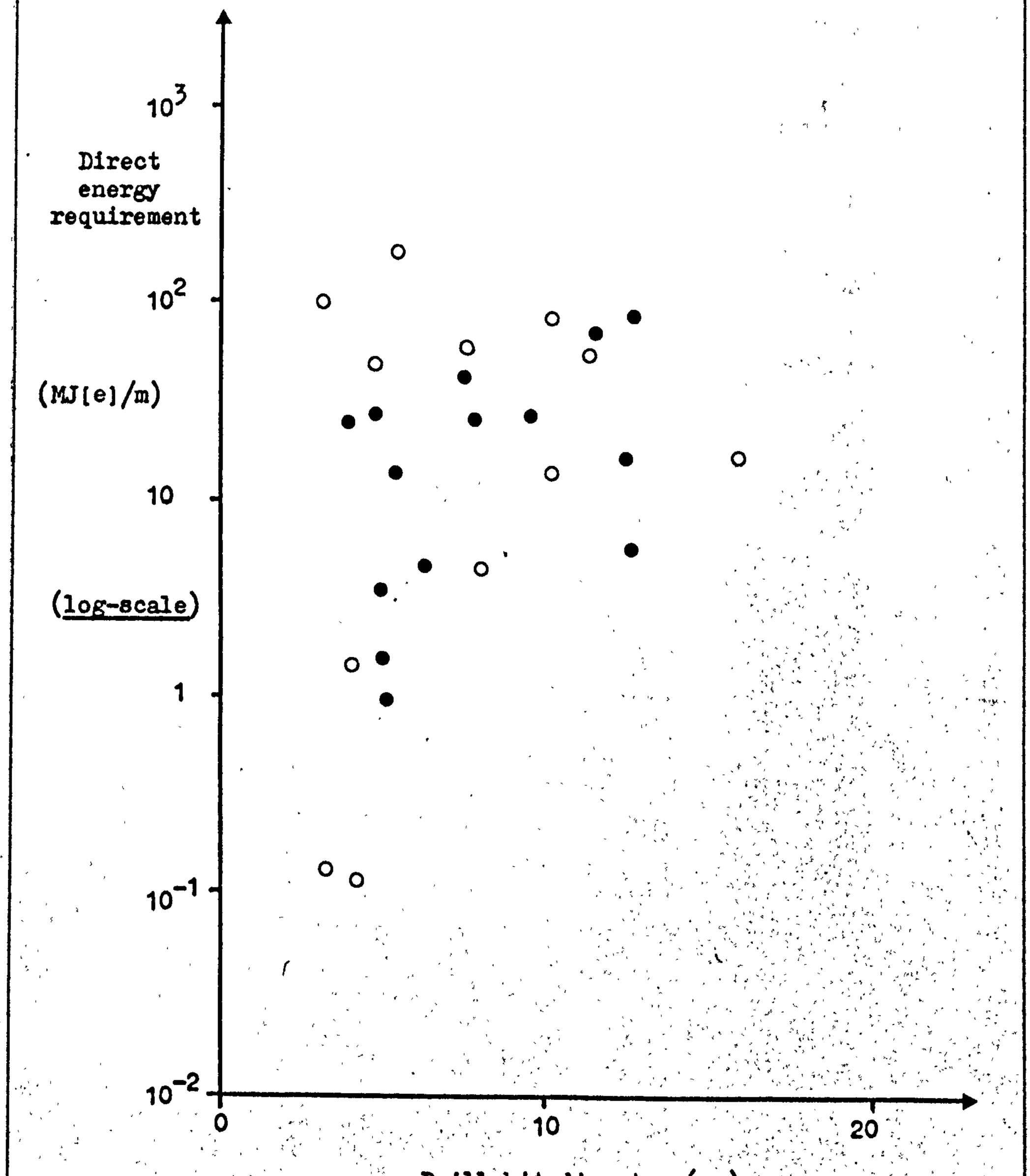


Table A.1 : Energy intensities of drilling equipment.(based on data from Casper, Chapman and Mortimer, 1974)

Item	Energy intensity (MJ/$: 1965-75)	
	electrical	thermal
Underground drills and jumbos :	3 ± 1	57 ± 18
Surface drill rigs :	3 ± 1	66 ± 24
Bits, shanks, rods and extras :	3 ± 1	65 ± 26

Table A.2 : Energy requirements of exploratory drilling.

Input	Energy requirement (MJ[t]/m)	
	minimum	maximum
Fuel :	0.6	1000
Drill bits and extras :	24.0	130
Drill rig and other equipment :	0.4	45
Maintenance :	-	25
TOTAL =	25.0	1200

Figure A.2 : Variation of the drilling required for blasting with rock fragmentation size.

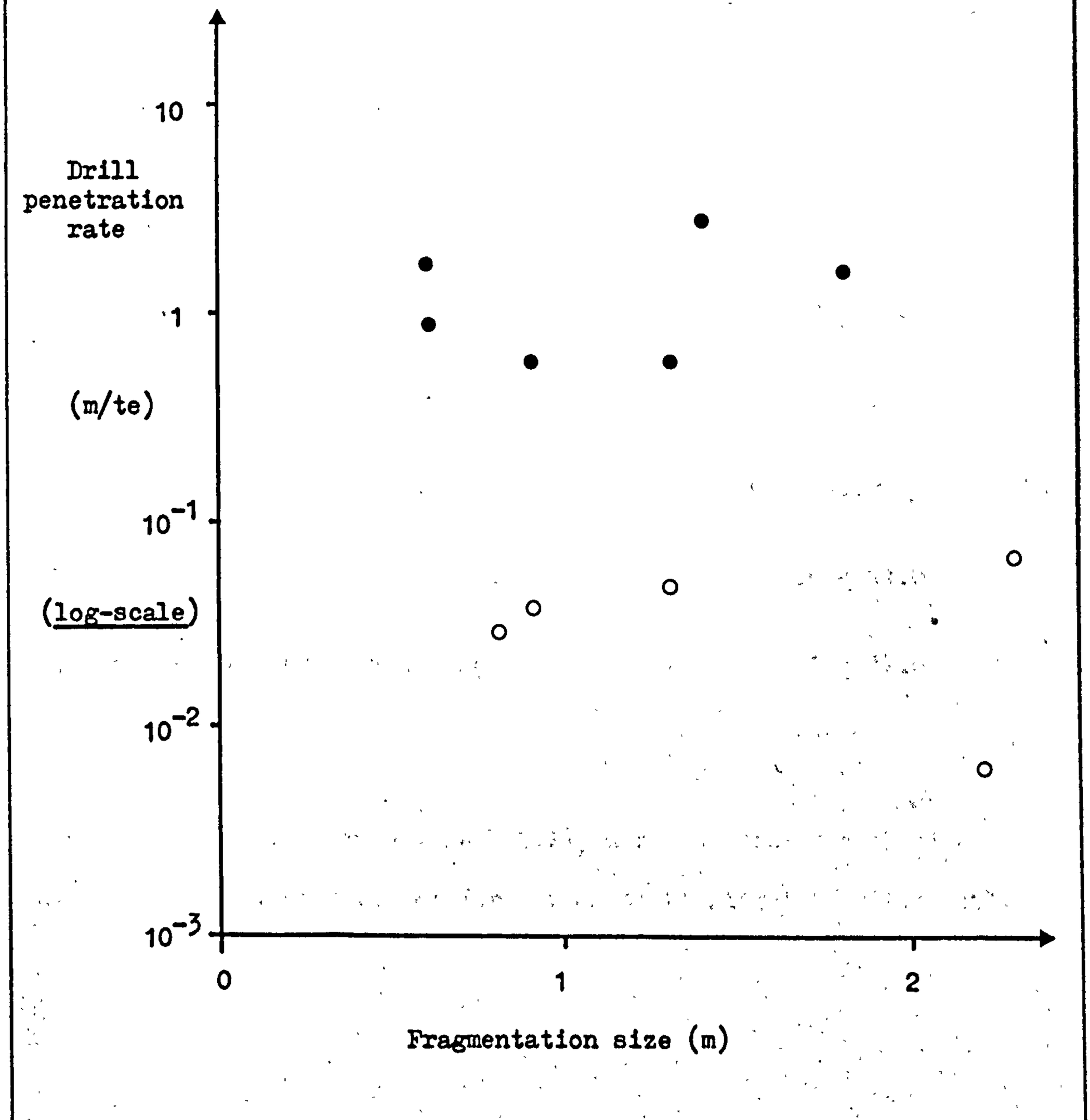
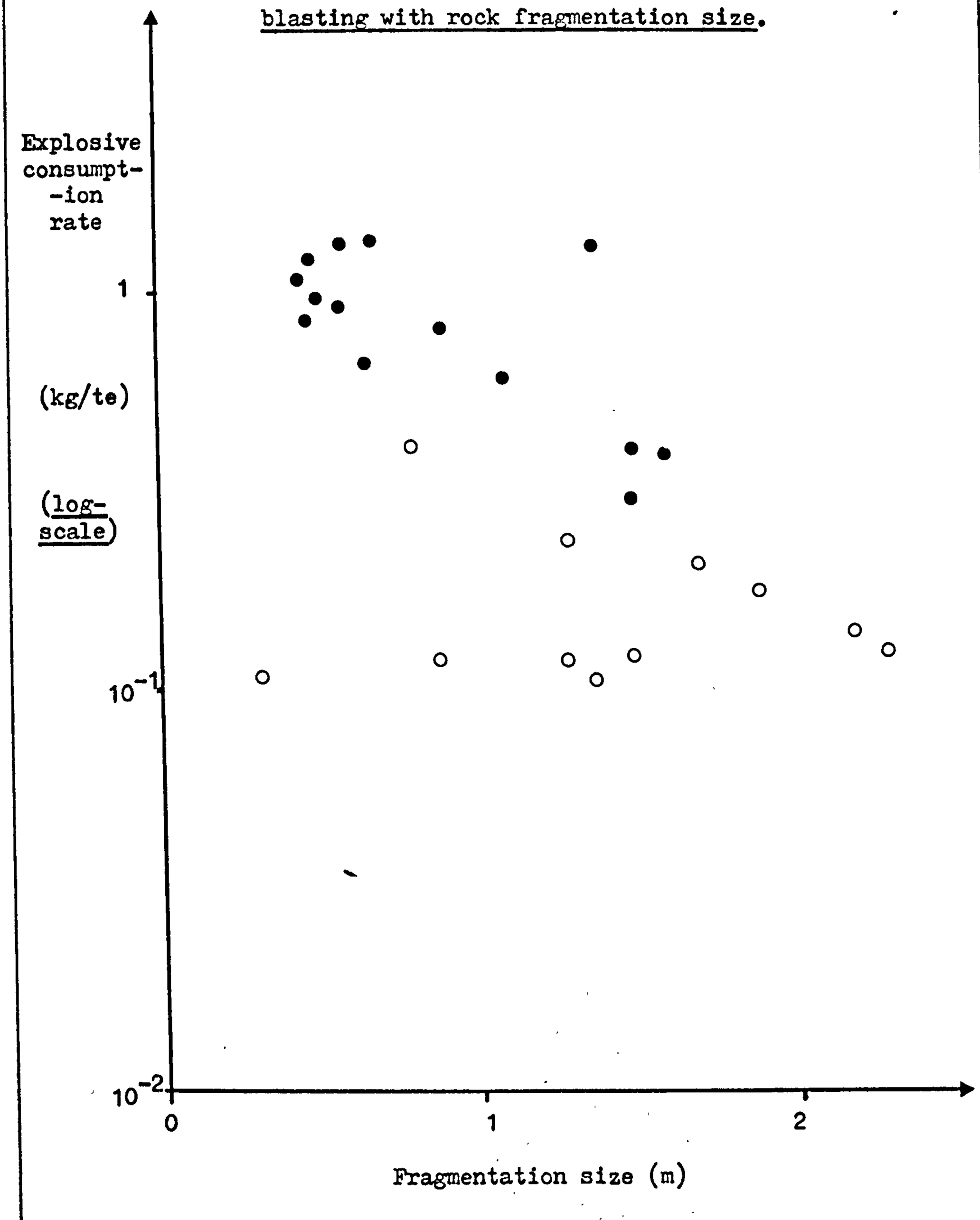


Table A.3 : Energy requirements of explosives. (based mainly on data from Casper et al, 1974)

Item	Units	Energy requirement (MJ)	
		electrical	thermal
Safety fuse :	metre	-	3.2 ± 0.8
Electric detonators :	each	-	12.5 ± 3.5
Blasting powder :	kg.	1	83.0 ± 19.0
Blasting caps :	each	-	3.5 ± 1.5
Ammonium nitrate* :	kg.	-	26.0 ± 6.0
Fuel oil* :	kg.	-	49.0
	litre	-	46.0
Gelatinous high explosive :	kg.	1	60.0 ± 11.0
Non-gelatinous high explosive :	kg.	1	53.0 ± 17.0

* for AN-FO (Ammonium Nitrate - Fuel Oil), a common mining explosive consisting of different proportions of ammonium nitrate and fuel oil.

Figure A.3 : Variation of the rate of use of explosives for blasting with rock fragmentation size.



KEY: ● = underground mining

○ = open-pit mining

Table A.4 : Energy requirements of underground drilling and blasting.

Item	Energy requirement (MJ/te) minimum	maximum
Fuel :	0.5 (e)	37.0 (e)
Drill bits and extras :	2.0 (t)	107.0 (t)
Drill rig and equipment :	5.0 (t)	18.0 (t)
Explosives :	15.0 (t)	107.0 (t)
Detonators and extras :	6.0 (t)	23.0 (t)
TOTAL =	0.5 (e) + 28.0 (t)	37.0 (e) + 255.0 (t)

Table A.5 : Energy requirements of surface drilling and blasting.

Item	Energy requirement (MJ/te) minimum	maximum
Fuel :	0.1 (e)	1.0 (e)
Drill bits and extras :	0.3 (t)	9.1 (t)
Drill rig and equipment :	1.0 (t)	8.9 (t)
Drill maintenance :	0.5 (t)	2.0 (t)
Explosives :	3.2 (t)	20.0 (t)
TOTAL =	0.1 (e) + 4.0 (t)	1.0 (e) + 40.0 (t)

Appendix B : Other mining activities

Following drilling and blasting, broken ore and other material is extracted from the deposit and moved from the workface to storage areas and waste dumps. Excavation and haulage consumes significant amounts of fuel in the mine, although additional, ancillary operations can also require large energy inputs. These other activities include mine services such as ventilation, heating and cooling, water drainage, etc., and the subsequent transportation of ore from the mine to the next stage in the nuclear fuel cycle, the ore processing plant, or mill. The total amount of energy used in all these operations was estimated by combining the results of the data base (see chapter 3) with information from reports describing a number of uranium mines (Barrett et al, 1958; Dare, 1959a, 1959b; Douglas, 1959; Soulé, 1959; Dare, 1960a, 1960b; Dare and Lindstrom, 1960; McCutcheon and Futterer, 1960; Nelson, 1960; Pearse, 1961; Everett, 1962; Olsson, 1964; Hurst, Crouse, Brown and Ross, 1966; Bieniewski, Persse and Brauch, 1971).

As with drilling and blasting, various basic factors such as ground conditions, type of mining, design and operation of equipment, etc., affect the energy input to these activities. This is reflected in the variation of the fuel consumption of different machines operating under various conditions in open-pit mines shown in figure B.1. The indirect 'capital' energy input of excavation and haulage equipment was deduced from the results of the data base and information about the working life of machinery.

The gross energy requirements of typical mining equipment produced in the UK are illustrated in table B.1. Estimates of the total amount of material handled by such equipment and subsequent indirect energy inputs were calculated assuming typical operating lives from 10^4 to 10^5 hours.

To evaluate the total energy input of these operations it is also necessary to investigate the consumption of other items such as lubricating oil, tyres and spare parts. An example of the effect of all such items on the total amount of energy required to excavate one cubic metre of earth with typical open-pit mining machinery is shown in figure B.2.

The direct and indirect energy requirements of miscellaneous activities are obviously influenced by such factors as the type of mining, working conditions, climate, etc., which can vary widely. A number of different mines were analysed to assess the extent of this variation and the results are exemplified by table B.2 which indicates the energy inputs to two particular uranium mines.

The energy input of transporting ore from mine to mill was deduced from transportation g.e.r. results, information about the distance travelled and estimates of the operational load factor. The g.e.r. of transportation depends on the type of vehicle used and figure B.3 illustrates the range of values available for typical freight operations. The figure shows the variation of g.e.r.'s with system capacity which indicates the rate

that freight can be handled. The distance between mine and mill is obviously determined by location and the current range for all the plants investigated was found to be from as little as 0.8 km to as much as 240 km. The actual amount of travelling involved in transportation was calculated by multiplying the mine-mill distance by the load factor, which is a measure of the freight capacity utilised. Since most freight vehicles travel to the mill full and return to the mine empty, the average load factor was assumed to be 0.5.

Figure B.1 : Direct energy requirements of typical medium-sized open-pit excavation machinery.

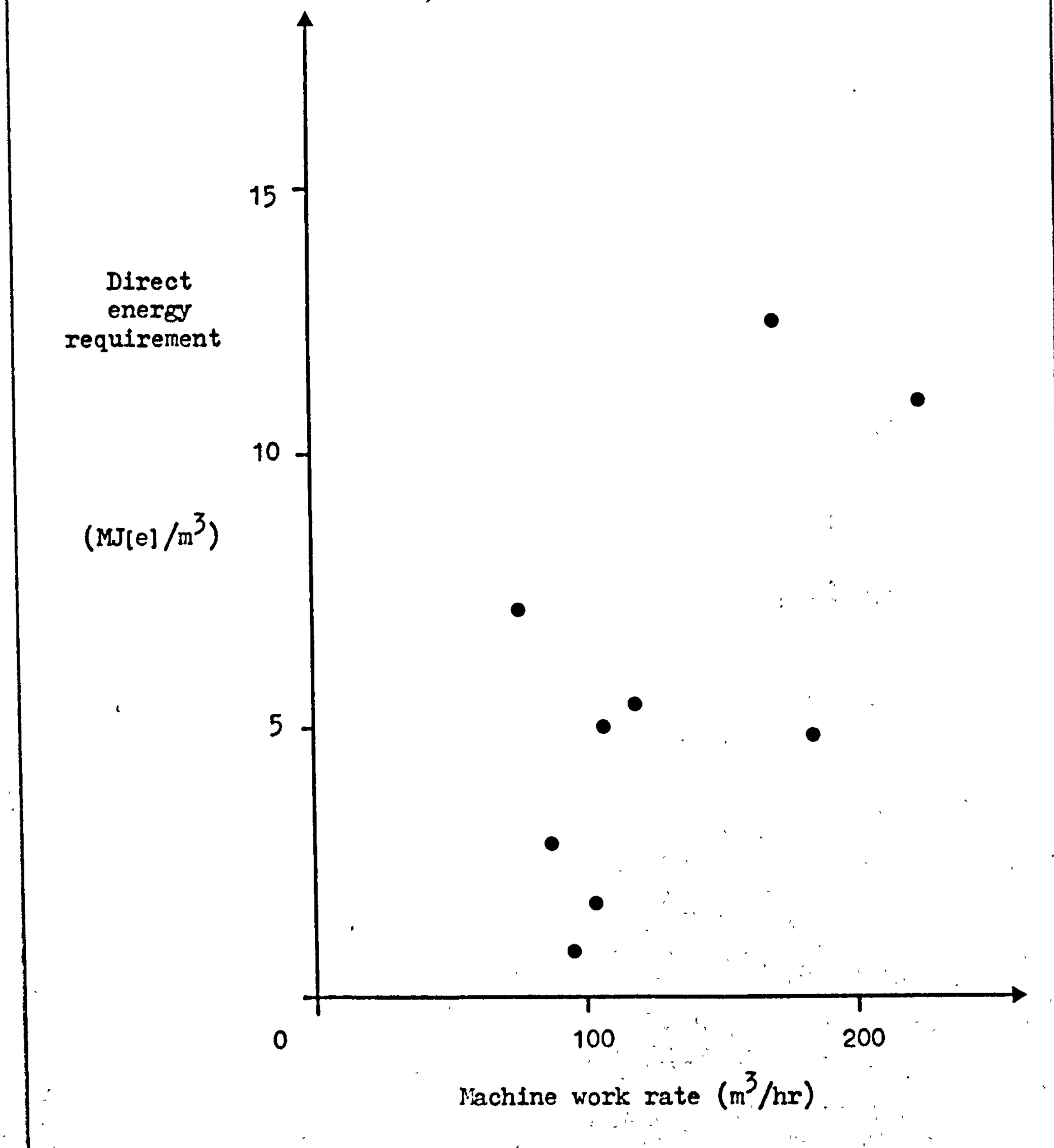


Table B.1 : Gross energy requirements of mining equipment. (based on data from Casper, Chapman and Mortimer, 1974)

Item	Gross energy requirement (10^6 MJ[t])
1 - 6 m^3 excavators :	1.6 ± 0.20
2 - 8 m^3 loading shovels :	1.4 ± 0.20
2 - 8 m^3 power shovels :	1.4 ± 0.70
150 - 450 kW crawler tractors :	1.4 ± 0.20
150 - 450 kW bulldozers :	0.2 ± 0.05
10 - 20 m^3 motor graders :	0.7 ± 0.10
300 - 800 te/hr conveyor belts :	1.1 ± 0.70
10 - 15 m^3 pit trucks	1.3 ± 0.50

Figure B.2 : Energy inputs to a typical motor-scraping excavation operation.

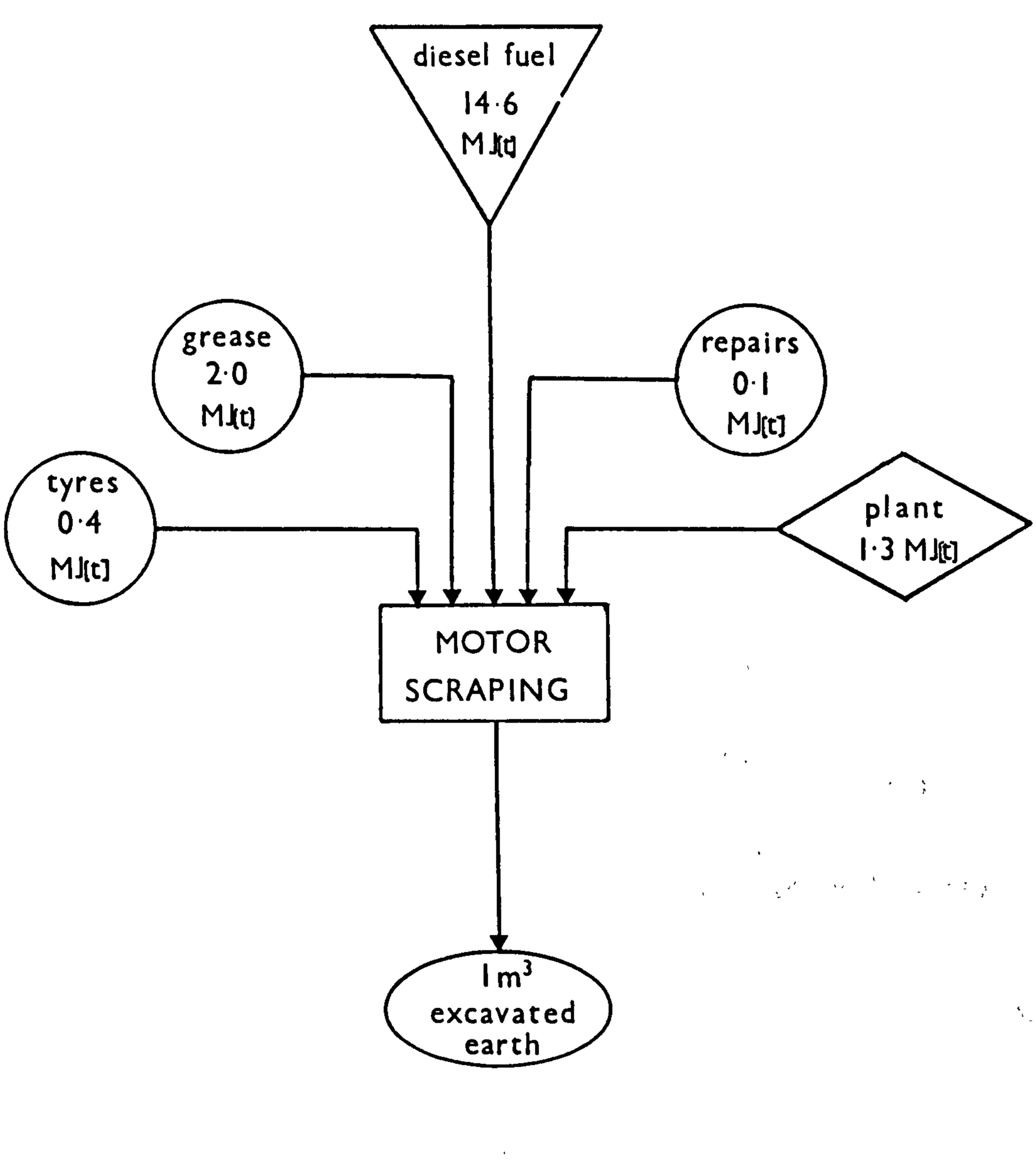
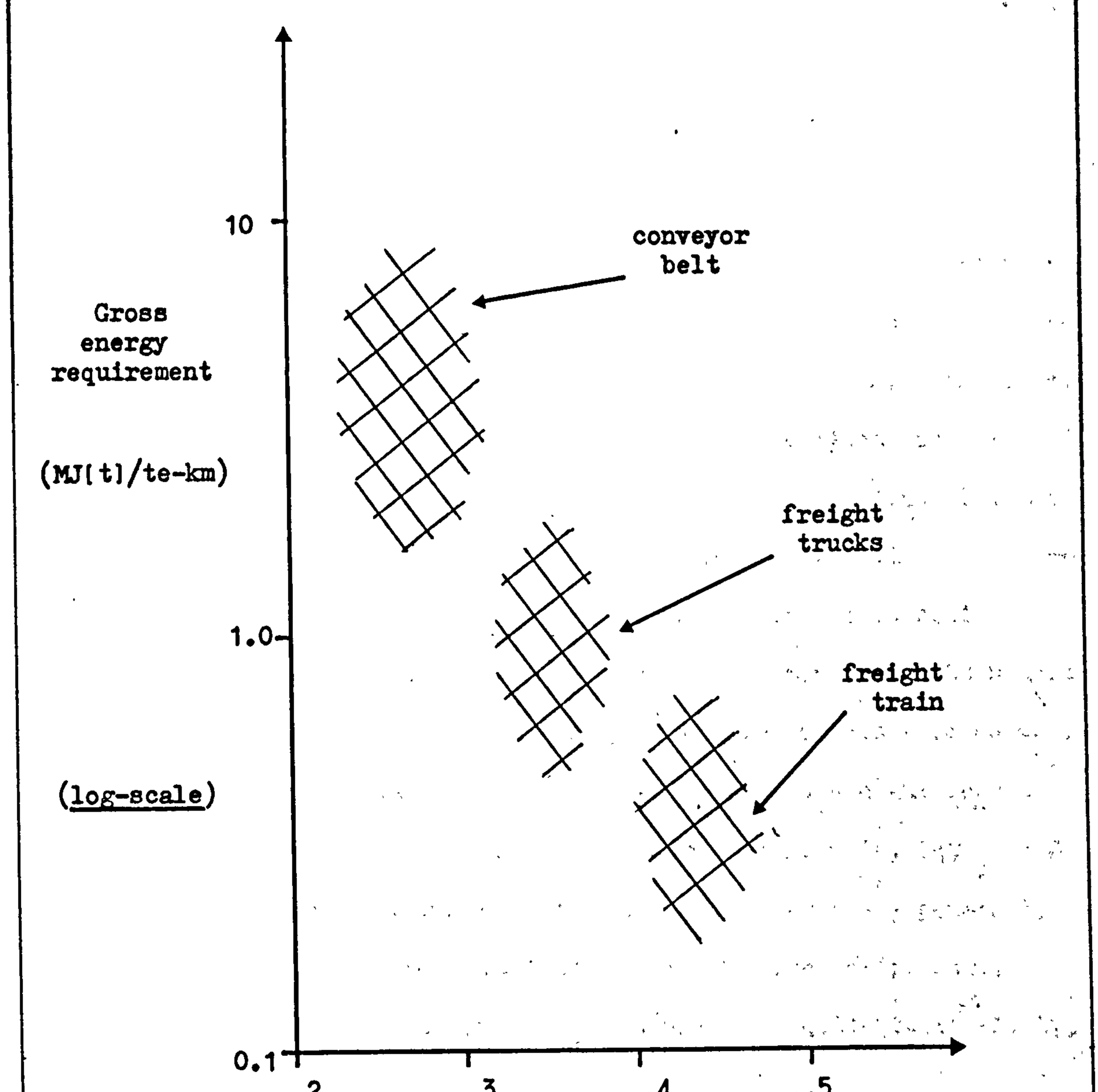


Table B.2 : Energy inputs to an underground and an open-pit uranium mine.

Item	Energy input (MJ/te material)	
	Canadian underground mine	U.S. open-pit mine
Drilling fuel :	17 (e)	0.5 (e)
Excavation fuel :	29 (e)	8.5 (e)
Water pumping fuel :	2 (e)	5.0 (e)
Ventilation fuel :	16 (e)	-
Heating fuel :	31 (t)	-
Stockpiling fuel :	-	6.0 (e)
Explosives :	1 (e) + 79 (t)	6.0 (t)
Plant and equipment :	5 (e) + 100 (t)	4.0 (t)
TOTAL =	70 (e) + 210 (t)	20.0 (e) + 10.0 (t)

Figure B.3 : Estimated variation of the g.e.r. of freight transport with system capacity.



Appendix C : Conventional treatment of uranium ores

The aim of this appendix is to describe the analysis of all common processes used to treat uranium ores. Specific details of processes involved in the production of uranium concentrates from natural materials are basically determined by the particular characteristics of the initial source of uranium in question. Consequently numerous methods which are currently in use or under development were investigated.

Many types of ore can be treated by conventional processing techniques and, for the purpose of analysis, these ores are divided into two distinct classes; orthodox and unorthodox. Orthodox ores are currently the most important sources of uranium, whilst unorthodox ores constitute potential sources for the future. There are two general categories of orthodox ore; acidic and alkaline. Acidic ores contain iron sulphide, FeS_2, and other pyritic minerals. Such ores include most of the commonly-mined uranium minerals as well as uranium-bearing granite, monazite sand and auriferous ores. Alkaline ores contain calcium carbonate, $CaCO_3$, and other acid-consuming materials. These ores are generally considered less important than acidic ores at present, although some large deposits are worked quite successfully.

Unorthodox ores consist of all the remaining primary ore types which are presently mined on a small scale or are expected to be used more widely in the future. These ores include carboniferous minerals such as low-grade uranium-bearing coals; lignites and bituminous shales as well as uraniferous leached-zone phosphate clays.

The conventional treatment of all these sources of uranium is basically similar. Ore from the mine is first crushed and ground to reduce the size of the material and expose uranium-bearing particles. Milled ore is then treated with appropriate leach reagents which dissolve uranium and the resulting mixture is thickened and filtered to produce a solid-free liquid. This is purified by various methods and an uranium-rich concentrate, often called wet cake, is obtained by adding suitable precipitating agents. This intermediate product is then dried to form a finished material, known as yellow cake.

Actual details of the processes, chemicals, equipment, etc., used depend on the particular type of ore under treatment. There are five important methods available for the treatment of acidic ores and these are the dilute-acid leach process, the strong-acid leach process, the sulphidic oxidation process, the microbiological leach process and the acid cure process.

The dilute-acid leach process is a traditional technique which is currently used in most uranium mills. This method involves the milling of ore with water to a fine size, leaching with relatively dilute sulphuric acid, purifying the leach liquor with ion-exchange resins or solvent extraction chemicals and precipitating uranium concentrate with ammonia, magnesium oxide, ammonia and calcium hydroxide, or calcium oxide and sodium hydroxide (Pinkney, 1956; Stuart, 1957; Clegg and Foley, 1958; Pinkney and Westwood, 1961; Everett, 1962; Mattson, 1967; Youngberg, 1973; U. S. Bureau of Mines, 1975)

The strong-acid leach process developed by the United Kingdom Atomic Energy Authority (UKAEA) consists of dry, coarse grinding and leaching with relatively concentrated sulphuric acid followed by treatment similar to the previously described dilute-acid process. Using this method, fuel consumption, wear and capital spending are reduced in the grinding section as well as savings in acid requirements and leaching times (Smith and Garrett, 1972; Lendrum, 1974; Engineering Mining Journal; 1975; Iammartino, 1975).

The sulphidic oxidation process avoids the need for conventional chemical leach reagents by utilising the presence of sulphide minerals in acidic ores. Using air under high pressure and continuous agitation, iron pyrites, FeS_2, contained in the finely ground ore, is oxidised to ferric sulphate, $Fe_2(SO_4)_3$, which acts as a substitute for sulphuric acid (Dasher, 1972; Smith and Garrett, 1972). Apart from reagent savings, it is also claimed that reductions in fuel consumption can be achieved with this method.

The microbiological leach process is a variation on the above technique. Oxidation of iron pyrites by bacteria such as Thiobacillus ferroxidans is used to leach the ore (McCreedy, Harrison and Gow, 1969; Gow et al, 1970; Bruynesteyn and Duncan, 1971). Although reagent consumption can be reduced by this method, further development work is still required to eliminate disadvantages such as leaching times which are generally five times longer than those of the dilute-acid process.

The acid cure process developed by Oak Ridge National Laboratories (ORNL) is an improved version of current methods and is intended for use in treating hard, low grade material such as granite (Hurst, Crouse, Brown and Ross, 1966). In this process ore is dry crushed into very coarse lumps which are cured in sulphuric acid. Leach liquor is washed from the ore and treated in a similar fashion to leach liquor from the dilute-acid leach process. Although this process can reduce milling requirements, the relatively low uranium recovery rate of about 66% can be a significant disadvantage.

Since acidic ores are the main traditional sources of uranium, acid leach processes, of one sort or another, have been used extensively throughout the nuclear fuel industry. Initially such processes were also used to treat alkaline ores (Commonwealth Mining and Metallurgical Congress, 1957; Clegg and Foley, 1958). However, calcium carbonate and similar compounds contained in these ores neutralise the leaching acid and this causes excessively high sulphuric acid consumption. Hence new techniques based on alkali leach reagents were developed, in particular the sodium carbonate leach process and the ammonium carbonate leach process.

With both these methods the ore is finely milled with water prior to leaching. In the sodium carbonate process, sodium hydroxide is used to produce a mixture of sodium carbonate and bicarbonate which leaches uranium from the ore. The resulting liquor is thickened and uranium is recovered by adding excess sodium hydroxide (Butler, 1951;

Forward, Halpern and Peters, 1953; Huttl, 1954; Hannay, 1956; Clegg and Foley, 1958; U. S. Bureau of Mines, 1975).

The ammonium carbonate pressure leach process was developed for alkaline ores containing a high proportion of silica which would react with sodium carbonate. High reagent consumption is avoided by leaching such ore with ammonium carbonate solution under pressure to prevent the decomposition of soluble uranium salts. After thickening and filtration uranium is precipitated by simply heating the leach liquor with steam (Langston, Macdonald and Stephens, Jr., 1957; Clegg and Foley, 1958).

In addition to acidic and alkaline ores, uranium can occur in minerals containing materials that can interfere with, or obviate the need for, certain operations in the above processes. Consequently, although processes for the treatment of orthodox ores have some aspects in common with previous methods, important differences do exist which depend on the specific characteristics of the ore in question. Three general classes of unorthodox ore are considered here; lignite, bituminous shale and leached-zone phosphate clay.

The first step in producing uranium from uraniferous lignites involves roasting this low grade fossil fuel to obtain a heavy metal-rich residue ash. Crushing and grinding can sometimes be avoided or reduced and, since the following uranium recovery is based on the treatment of ash in particular, wastes from furnaces burning uranium-rich coal and oil can also be processed. After

the ash has been slurried with water it is treated like an acidic ore with techniques similar to the dilute sulphuric acid leach process (Clegg and Foley, 1958; Mitchell, 1965).

Bituminous shales generally contain less carbonaceous material than lignites and consequently they are not usually burned to ash, as in the above process, prior to treatment. Oil can be extracted from the shale by retorting but unless conditions are carefully controlled the leachability of uranium from the resulting residue can be severely impaired. Hence processes have been developed primarily to recover uranium rather than oil from such shales and these incorporate techniques similar to those used in the processing of acidic ores (Swerke, 1955; Clegg and Foley, 1958; Bieniewski, Persse and Brauch, 1971; Andersson and Olsson, 1975).

The final source of uranium considered here is leached-zone phosphatic clay which overlies many phosphate rock deposits. Although uranium often occurs in association with phosphatic material, such minerals are usually regarded as important sources of phosphate fertilizers and chemicals rather than uranium ores. Hence uranium is generally recovered as a by-product from the phosphoric acid processing of phosphate rocks (see appendix D).

Uranium could be produced as a main product from material which contains only small amounts of phosphate such as leached-zone phosphatic clay. Three processes have been developed to treat this material. In the first two methods, the ore is dry ground and calcined, or simply wet

ground, then the uranium is digested with sulphuric acid and reprecipitated with a mixture of sulphuric acid and hydrofluoric acid. Quite pure uranium tetrafluoride, UF_4, or green salt, is obtained by these methods. The third technique involves dry grinding followed by calcination, nitric acid digestion and ammonia precipitation to produce ammonium diuranate (Clegg and Foley, 1958; Bieniewski et al, 1971).

All the processes described above are being used or could be used on a commercial scale to extract uranium from quite common ores. Consequently all were investigated to determine the energy requirements of the conventional processing of uranium ore. A number of new methods were not examined in detail either because they have limited use or because little information is available about them. One particular process that has recently received much attention but was not included in this analysis consists of standard acid leaching followed by chlorination, electrolysis and hydrofluorination to produce uranium tetrafluoride (Bodu, 1971; Iammartino, 1975). Despite eliminating several steps in the fuel cycle there are indications that this technique may be restricted to only certain types of ore.

Energy analysis of ore processing involved examining research papers, plant operating reports and company data. Many individual processes were investigated and flowcharts, such as the examples shown in figures C.1, C.2 and C.3, were used to deduce the total energy input. All operations in processing were studied carefully to

assess the full extent of variations in the energy requirement. In particular, literature describing comminution - crushing and grinding - was examined in detail (eg. Lowrison, 1974), since such processes can consume substantial amounts of fuel. The effect of ore characteristics on reagent consumption (Clegg and Foley, 1958) was also assessed - see figure C.4 and C.5.

For the purpose of analysis three particular results describing fundamental aspects of ore processing were evaluated. These are the energy requirement of producing wet uranium concentrate, or 'wet cake', from ore, the energy requirement of drying wet cake and the overall uranium recovery efficiency. The energy requirement of wet cake production comprises energy inputs to all stages from ore crushing and grinding to uranium precipitation. The energy requirement of wet cake drying also includes the energy used in the handling and packing of concentrates. The overall uranium recovery efficiency relates total uranium input, in the form or ore, to final output, as dry concentrate, or yellow cake.

Table C.1 illustrates wet cake production energy requirements for different conventional processing techniques. Differences in ore characteristics are largely responsible for the minimum - maximum variation shown and basic dissimilarities in processing are reflected in differences between e.r. values. The results indicate that the energy required to produce wet cake from one tonne of any conventional ore ranges from about 300 MJ(t) to 10000 MJ(t), assuming an electricity conversion factor of 4 MJ(t) per MJ(e).

The maximum energy requirement of drying wet cake for various ore processing methods is shown in table C.2. In certain climates it is possible to dry concentrates using solar heating and consequently it is possible to achieve a minimum fuel input corresponding to the fuel consumption of yellow cake handling and packing. Hence the energy required to dry wet cake lies in the range 5 MJ(t) to 552000 MJ(t) per tonne triuranium octoxide, U_3O_8.

The relative efficiency of each process investigated here is given in table C.3. Although most methods can recover a substantial proportion of the uranium available in the ore, the acid cure process for granites and the leached-zone phosphate clay process are noticeably less efficient. Hence the possible extent of recovery rates is from 0.42 and 0.98.

Figure C.1 : Processing with ion-exchange and Resin-In-pulp recovery for a very hard, acidic ore.

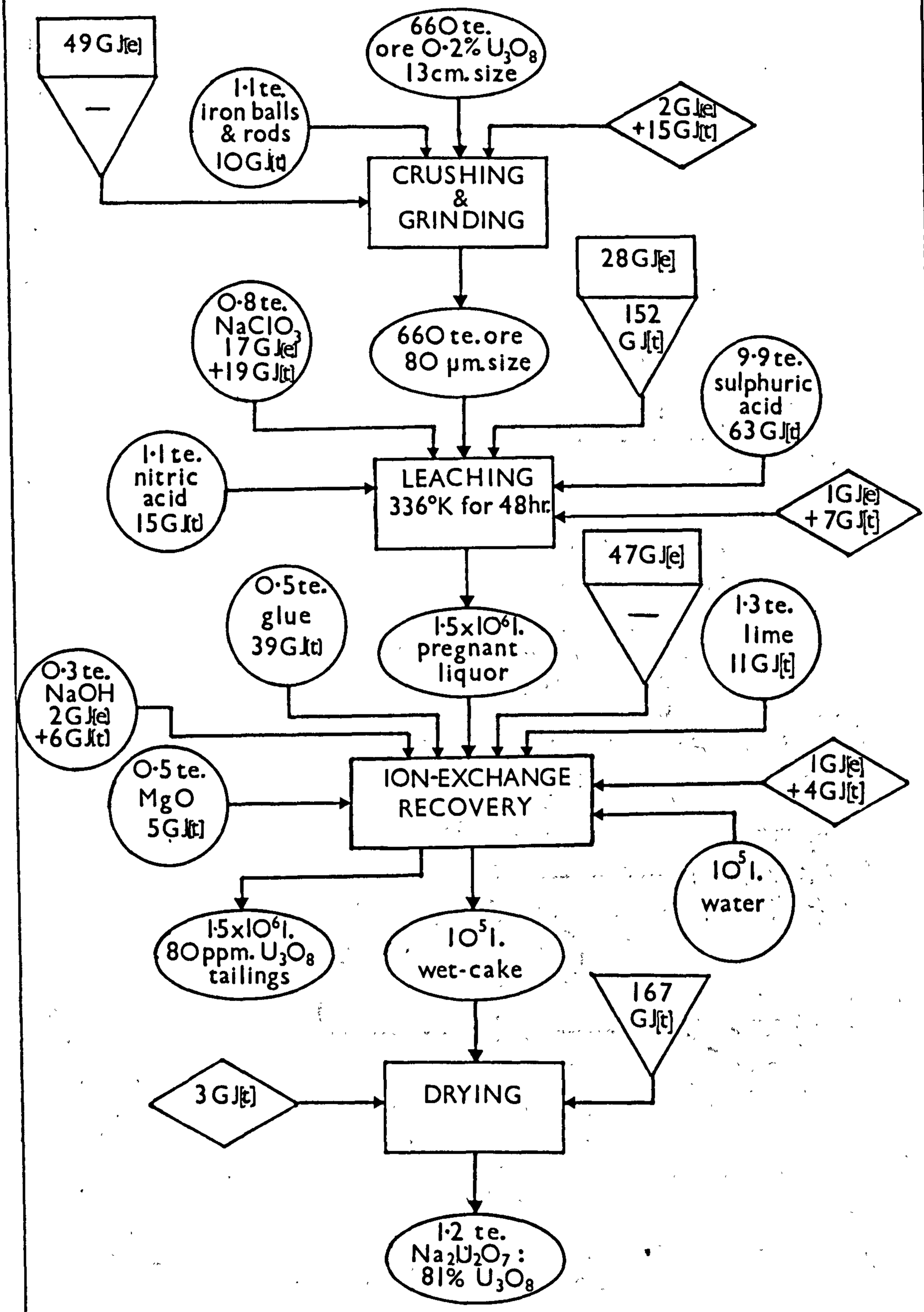


Figure C.2 : Processing with solvent extraction recovery for a soft, alkaline ore.

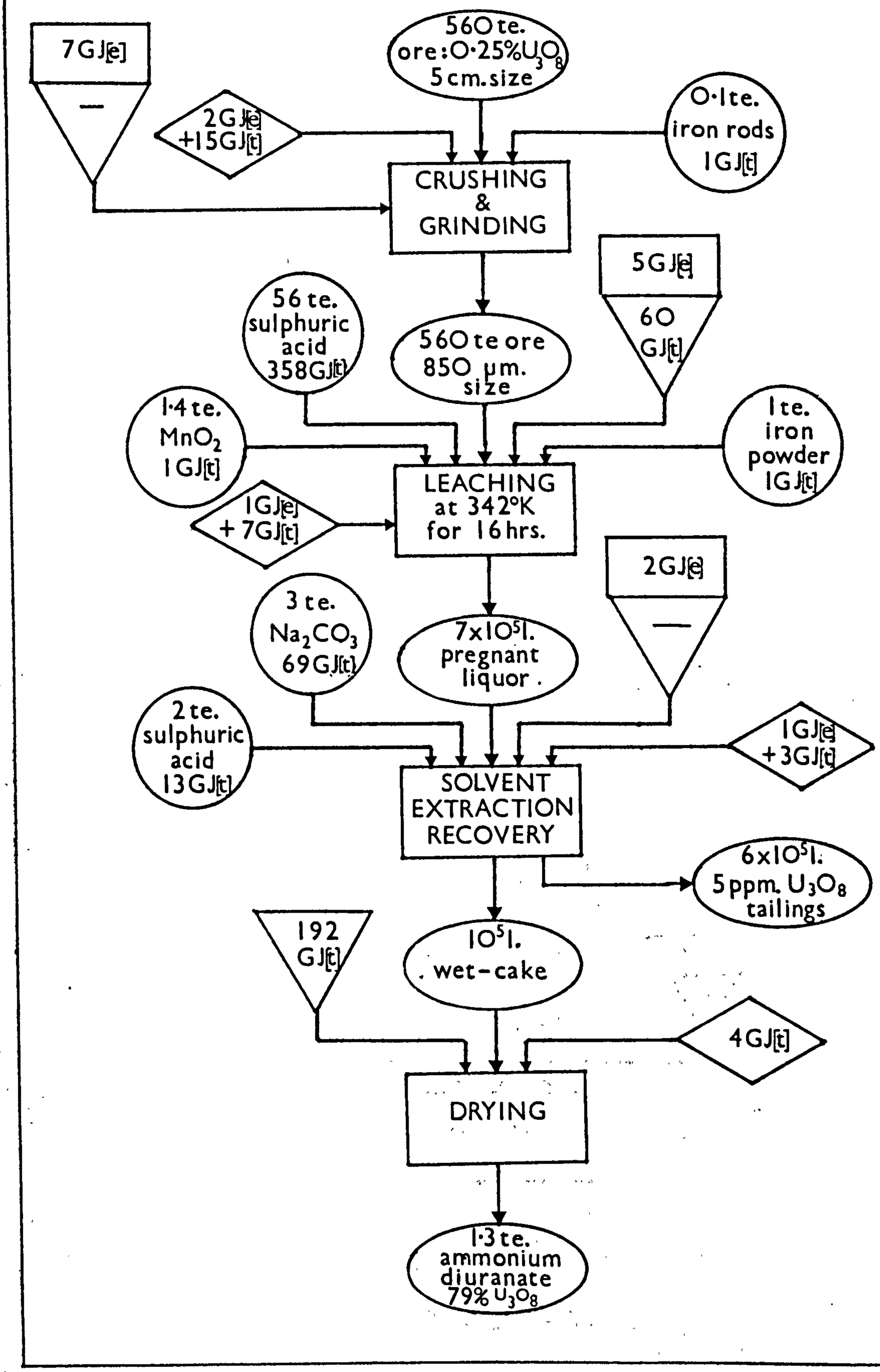


Figure C.3 : Processing with ion-exchange and Resin-In-Pulp recovery for a very hard, alkaline ore.

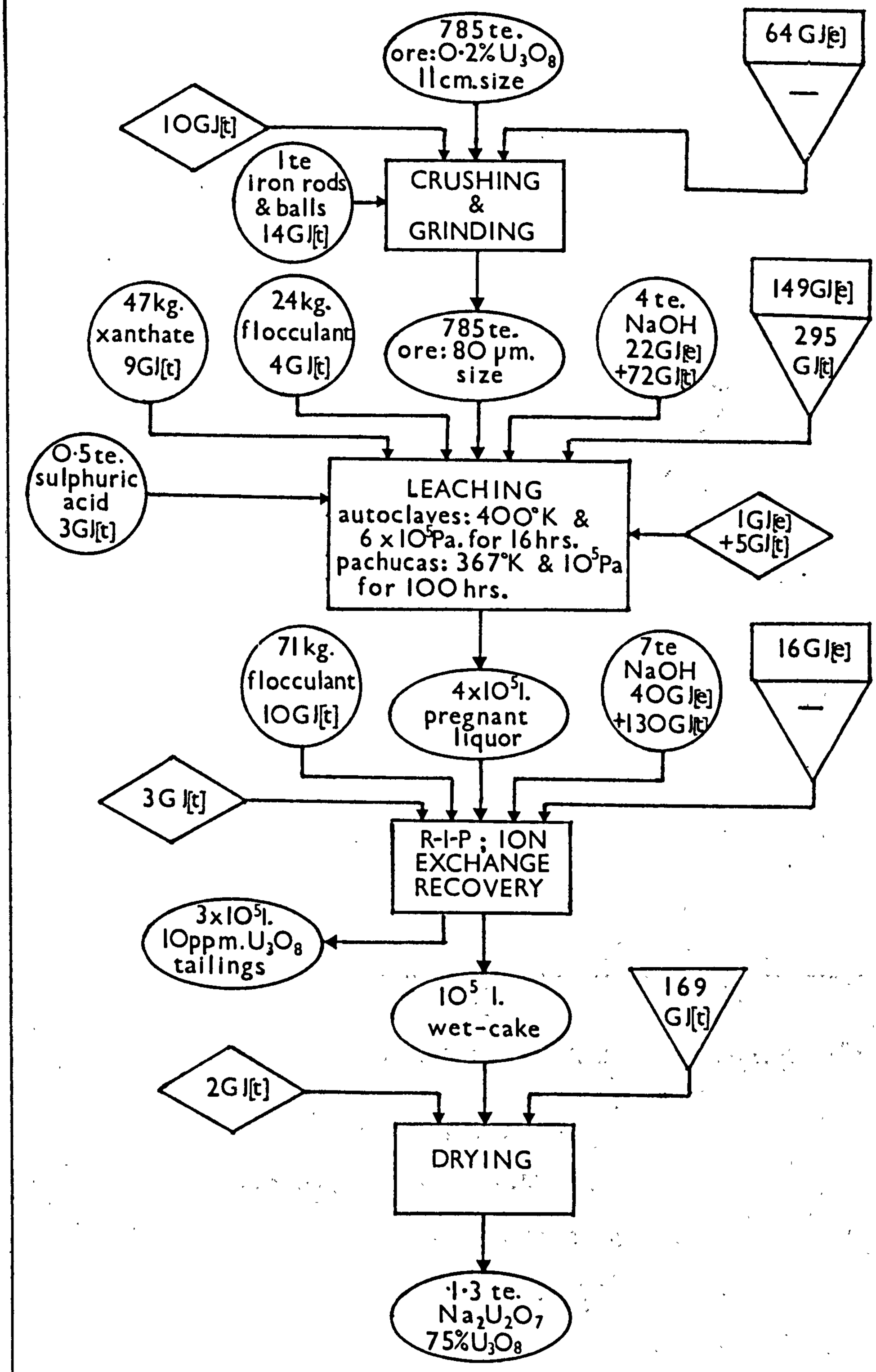
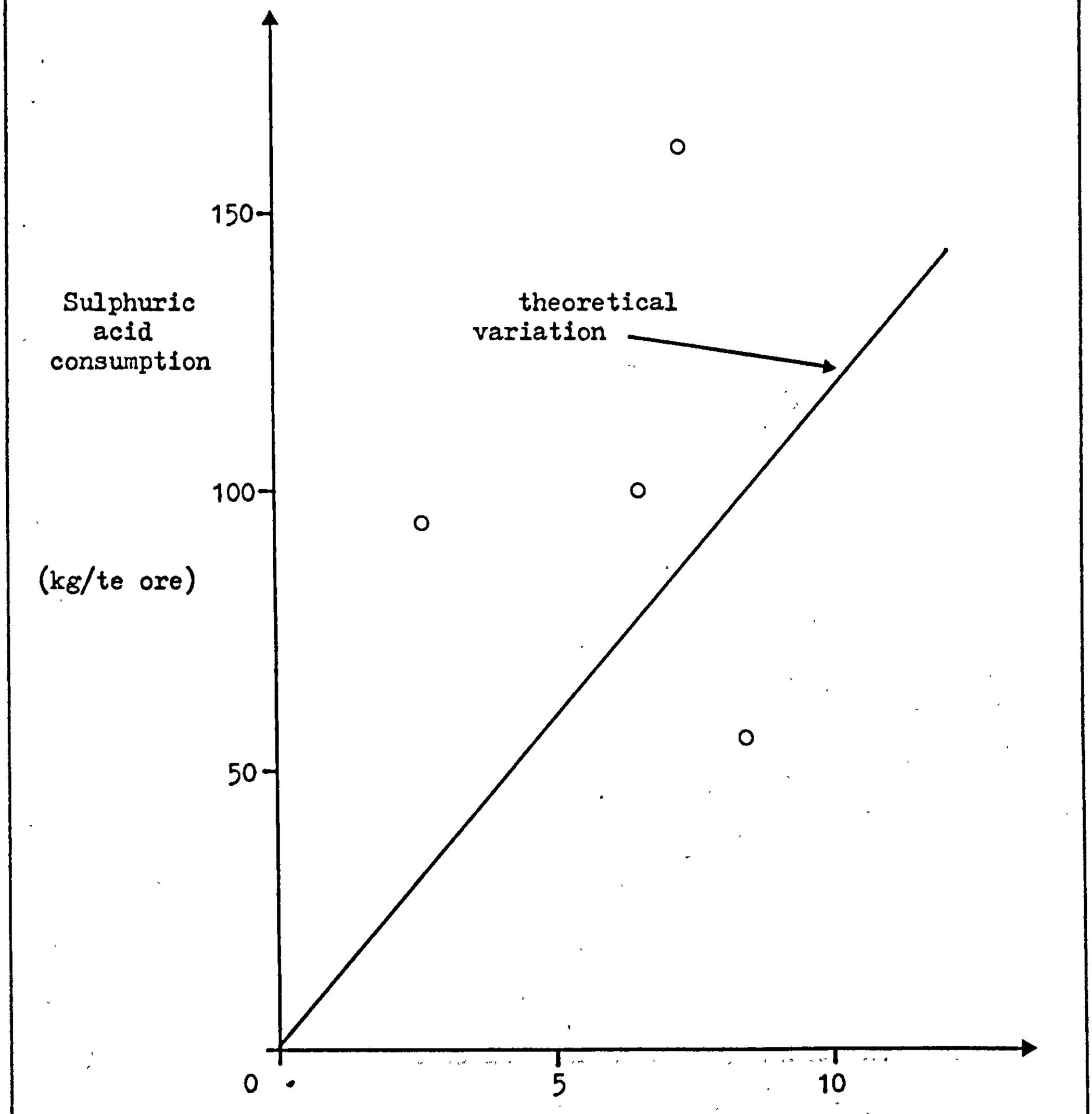


Figure C.4 : Effect of ore alkalinity on acid leach reagent consumption.



o = observed data point

* assumes; 1% $CaCO_3$ ≡ 0.55% CaO

≡ 0.34% Na_2O

≡ 0.45% K_2O

Figure C.5 : Effect of ore acidity on alkali leach reagent consumption.

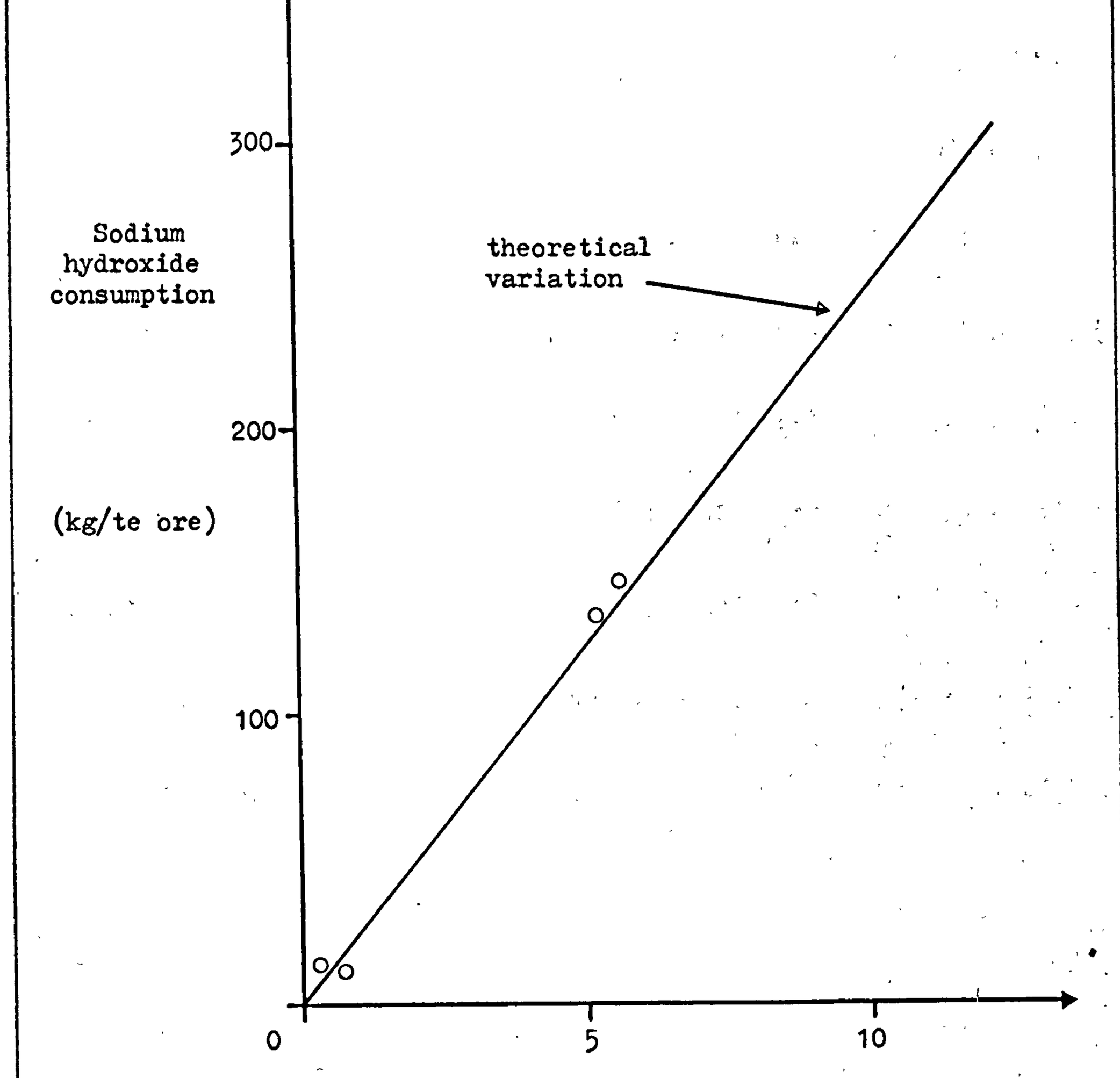


Table C.1 : Energy requirements for the production of wet cake from ore.

Process route	Energy requirement (MJ/te ore) minimum	maximum
Dilute acid leach :	50 (e) + 250 (t)	295 (e) + 3040 (t)
Strong acid leach :	60 (e) + 225 (t)	285 (e) + 2160 (t)
Sulphidic oxidation :	85 (e) + 240 (t)	350 (e) + 1520 (t)
Microbiological leach :	510 (e) + 195 (t)	1610 (e) + 3650 (t)
Acid cure :	35 (e) + 130 (t)	70 (e) + 865 (t)
Sodium carbonate leach :	100 (e) + 835 (t)	395 (e) + 1220 (t)
Ammonium carbonate leach :	135 (e) + 1200 (t)	325 (e) + 1470 (t)
Lignite treatment :	45 (e) + 150 (t)	120 (e) + 5530 (t)
Bituminous shale treatment :	40 (e) + 530 (t)	95 (e) + 1900 (t)
Phosphate clay treatment :	135 (e) + 980 (t)	165 (e) + 2300 (t)

Table C.2 : Maximum energy requirements for the drying of wet cake.

Route	Energy requirement (MJ[t]/te U_3O_8)
Acidic ore processing :	188000
Alkaline ore processing :	196000
Lignite processing :	192000
Bituminous shale processing :	225000
Phosphate clay processing :	552000

Table C.3 : Uranium recovery efficiencies.

Process route	Recovery factor
Dilute acid leach :	0.85 - 0.95
Strong acid leach :	0.94 - 0.96
Sulphidic oxidation :	0.95
Microbiological leach :	0.90
Acid cure :	0.66
Sodium carbonate leach :	0.80 - 0.93
Ammonium carbonate leach :	0.98
Lignite treatment :	0.88
Bituminous shale treatment :	0.70 - 0.80
Phosphate clay treatment :	0.42

Appendix D : Unconventional treatment of uranium ores.

In addition to the common techniques of ore processing discussed in appendix C, a number of unconventional methods for the production of uranium from orthodox and unorthodox ores were also investigated. These included a technique for the recovery of uranium from deposits which avoids the usual mining and milling operations, a method for treating uranium-bearing phosphoric acid and a process for extracting uranium from the leach solution of copper processing mills. All three techniques are described in this appendix and the energy requirements of each are evaluated.

The process which enables uranium to be extracted from the ground and separated from gangue without conventional mining and milling is known as 'in situ' leaching. This involves drilling a number of feed wells into the deposit through which a reagent such as sulphuric acid is pumped. Uranium is leached from the ore and pregnant liquor containing between 50 and 100 parts per million (ppm) is recovered from suitably placed production wells. Common methods of thickening, clarification and precipitation are then used to obtain yellow cake (Mining Magazine, 1971b).

Although the technique of in situ leaching is still under development, it is claimed to have a number of advantages over conventional mining and processing such as short lead times, low capital cost, reduced labour requirements and negligible waste disposal charges (Marrs, 1970). However, serious problems currently restrict the application of this method. With present technology in situ leaching can only be used on ore deposits which are highly-porous and

yet hydrologically confined. Very little ore can be leached from compact deposits containing few fractures whilst too much reagent can be lost from deposits surrounded by very permeable strata. Even with reasonable geological conditions the amount of uranium that can be extracted from the deposit can be low and current recovery rates are about 20%. The most common use of this method is 'heap leaching' which involves recovering uranium from the low-grade waste or tailings of conventional mills. Such techniques accounted for approximately 1% of US uranium concentrate production in 1971 (U.S. Atomic Energy Commission, 1972).

As regards energy analysis, in situ leaching can be divided into three distinct operations; drilling feed and production wells, pumping leach reagent through the system and treating pregnant liquor to obtain uranium concentrate. To deduce the range of energy inputs, operations under both the best and worst conditions were investigated. For the best conditions it was assumed that the ore and surrounding overburden was relatively soft and friable which results in low drilling energy requirements, the well life was long - 6 months is the longest experienced - and the leach reagent consisted of used or 'barren' solution from a neighbouring mill. The worst conditions were characterised by hard, compact rock giving high drilling energy requirements, low well life due to silting of about 1 month, and the use of fresh leach reagent that is easily lost through bad drainage.

Minimum and maximum values for the total energy input to in situ leaching were evaluated using these basic

assumptions. Results are shown in table D.1 which also illustrates the range of g.e.r.'s for producing yellow cake from ore with conventional mining and traditional acid leach processing (see appendix C). Comparison shows that, on average, in situ leaching consumes almost 50% less energy than current methods.

Ores other than those described in appendix C have been considered, at one time or another, as significant sources of uranium and consequently processes have been designed and developed for their treatment. In particular, techniques for the recovery of uranium from certain types of phosphate minerals and copper ores are discussed here. Although the ore grades of uraniferous phosphate rock, around 0.01 to 0.02% U_3O_8, is generally much lower than current commercial levels, resources of such material appear to be quite substantial. US phosphate deposits, for example, amount to roughly 2×10^6 tonnes U_3O_8 which accounts for 20% of the current world uranium resources of ore with grades higher than 50 ppm U_3O_8 (see appendix F). Uranium-bearing copper ores could offer similar amounts of uranium with a possible world resource base of 2×10^6 tonnes U_3O_8 and a US supply of 3×10^4 tonnes U_3O_8 (Bieniewski, Persse and Brauch, 1971). Such ores, however, are very low grade, averaging 25 ppm U_3O_8, and resources are insignificant compared to similar grade material amounting to 3×10^9 tonnes U_3O_8.

Uraniferous phosphate rocks and copper ores are secondary sources of uranium. These minerals are primarily treated for the recovery of phosphates and copper, respectively, and uranium is usually produced afterwards as a by-product.

Table D.1 : Energy requirements for the production of uranium from various sources by different methods (G = ore grade in % U_3O_8).

Route	Energy requirement (MJ/te U_3O_8)	
	electrical	thermal
Conventional mining and dilute acid leach treatment of orthodox ores - minimum :	$\frac{(5400)}{G}$	+ $\frac{(26500 + 5)}{G}$
- maximum :	$\frac{(370000 + 390)}{G}$	+ $\frac{(550000 + 190000)}{G}$
In situ leaching of orthodox ores - minimum :	$\frac{(7800)}{G}$	+ $\frac{(6700 + 5)}{G}$
- maximum :	$\frac{(21500 + 390)}{G}$	+ $\frac{(830000 + 200000)}{G}$
Wet-process phosphoric acid treatment - minimum :	$\frac{(3500)}{G}$	+ $\frac{(55000 + 120000)}{G}$
- maximum :	$\frac{(3500 + 820)}{G}$	+ $\frac{(111000 + 130000)}{G}$
Copper leach solution treatment - minimum :	$\frac{(330)}{G}$	+ $\frac{(260 + 47600)}{G}$
- maximum :	$\frac{(800 + 580)}{G}$	+ $\frac{(800 + 74000)}{G}$

Consequently methods designed to extract uranium from these ores are largely dependent on initial processing techniques.

A number of processes have been developed for the recovery of uranium during the production of phosphate chemicals (Clegg and Foley, 1958; Bieniewski et al, 1971; Chemical Engineering, 1975). One apparently successful method is the wet-process phosphoric acid process. Freshly-produced phosphoric acid contains approximately 100 ppm U_3O_8 and uranium is extracted from this cooled acid by an absorbing reagent, such as alkyl pyrophosphate ester, dissolved in an immiscible organic solution, typically kerosine. The uranium is recovered by adding dilute hydrofluoric acid to precipitate uranium tetrafluoride, UF_4. This 'green salt' has a high moisture content and it must be treated by centrifuges and dryers to produce a final product which contains 50% equivalent U_3O_8.

The energy required by this process was deduced by investigating flowchart operating data (Bieniewski et al, 1971). Uranium is a by-product because there is much less of this material than phosphate in the ore. Therefore, only operations directly contributing to the recovery of uranium were included in the energy analysis. Since mining, comminution and phosphate leaching operations are only associated with phosphoric acid production, subsequent costs and energy inputs are avoided by the by-product treatment. This results in lower energy requirements, as indicated in table D.1, compared with those of conventional processing of orthodox ores.

Although this method of producing uranium appears to be competitive, in financial and energy terms, with more

common techniques, it has a number of important disadvantages that are expected to restrict its wide-scale use. At the moment, the wet-process method is only one of many techniques used to treat phosphate minerals. In the USA wet-process phosphoric acid production only accounted for 25% of total phosphate manufacture during the period 1965 to 1968 (Bieniewski et al, 1971). Most of the remainder was supplied by the elemental phosphorus process or by the electric furnace phosphorus route (U.S. Bureau of Mines, 1972). Since there are currently no techniques for the recovery of uranium from these processes, their continued use represents a loss of potential nuclear fuel resources.

Even if uranium recovery methods could be developed for all phosphate processes, nuclear fuel supply would be dependent on the level of phosphate chemicals demand. At present the world phosphate industry could only provide 5×10^3 tonnes U_3O_8 each year (von Kienlin, 1976), which is about a quarter of current nuclear fuel requirements (Nuclear Energy Agency, 1975), and there are indications that uranium demand will further exceed the production of uraniferous phosphates in the near future (Emler, 1976; Times, 1976). If such minerals were treated as primary sources of uranium, production could more easily respond to nuclear fuel demand. However, in that case the financial advantages of the process would be eroded since uranium production alone would then incur the full costs of mining, milling, etc.

The copper leach solution process is a method that has been developed to recover uranium from cupriferous ores

(Bieniewski et al, 1971). Copper can be produced from certain ores by leaching followed by precipitation with iron and the resulting waste leach liquor has been found to contain between 1 and 2 ppm U_3O_8. Uranium can be extracted from this solution by passing it over anion-exchange resins which selectively absorb uranium. These resins are then regenerated with dilute sulphuric acid and uranium is subsequently extracted with an organic solvent. So-called loaded strip liquor is formed, from which ammonium diuranate is recovered by adding ammonia. This wet precipitate is then dried with heaters and cyclones to obtain a finished, low moisture content concentrate.

The energy required by this process was deduced from pilot plant operating data (Bieniewski et al, 1971) and the results are shown in table D.1. Comparison indicates that the copper leach process consumes, on average, only 1% of the energy of conventional techniques. Such low fuel consumption arises because the recovery process is a by-product technique (typical ore content is 0.6% copper and 25ppm U_3O_8) and , in common with the wet-process phosphoric acid method, avoids fuel contributions from energy-intensive operations such as mining and milling. This convention of discounting the costs and inputs of the main product process when analysing by-product methods was simply adopted from standard financial accounting procedure.

Although total costs reflect low energy consumption, it seems unlikely that the copper leach solution process will become a major source of uranium. Ore leaching is only

used to provide a small amount of copper - in the USA during 1971 just 10% of copper output was produced by this method (U.S. Bureau of Mines, 1972) - and uranium recovery techniques are not available for the more common flotation, concentration and smelting production route. If new recovery methods could be developed all the world's copper mills could provide about half the current nuclear fuel demand. Uranium supply, however, would be determined by copper demand. Hence, although the copper leach solution process may be a relatively cheap way to produce uranium, this route is not expected to contribute significantly to nuclear fuel supply and production is unlikely to exceed 10^3 tonnes U_3O_8 per year in the near future (Bieniewski et al, 1971).

Appendix E : Extraction of uranium from seawater

Previous appendices have described various processes for the production of uranium from land-based sources. Proposed methods for the recovery of uranium from seawater are considered here and the purpose of the appendix is to evaluate the energy required by these techniques.

Uranium occurs in seawater in the form of the tricarbonato uranyl complex ion, $UO_2(CO_3)_3^{4-}$, at a relatively uniform concentration of 3.3 parts per billion or 3.3 ppb U (Wilson et al, 1960). Seawater has frequently been suggested as an attractive source of uranium since the oceans appear to provide an easily-accessible, vast reservoir of this material (eg. Greenfield, 1970; Lewis, 1972). It has been estimated that the world's oceans contain an amount equivalent to almost 5×10^9 tonnes U_3O_8 (Haigh, 1974). This is substantially more than current reserves of commercial ores amounting to approximately 4×10^6 tonnes U_3O_8 (Nuclear Energy Agency, 1975), but less than the estimated resources contained in low-grade land-based rocks of 10^{14} tonnes U_3O_8 (Lewis, 1972).

The basic problem of obtaining uranium from seawater consists of selectively extracting significant amounts of uranium from a weak solution of tricarbonato-uranyl ions that also contains many other dissolved minerals and elements. Numerous processes have been proposed but the technique which is currently receiving much attention involves absorption, followed by re-dissolving, or eluting, and recovery of uranium concentrate by precipitation. Details of the extraction cycle depend on the type of absorber used and numerous absorbers have already been

tested, ranging from common lead ore, or galena, to cultured green algae (Laskorin, Metalnikov and Terentiev, 1958; Davies et al, 1964; Koyanaka, 1970; Heide, Wagner, Paschke and Wald, 1973). Although development work still continues, interest has recently centred around the use of titanium hydroxide as a practical absorber.

Titanium hydroxide, H_4TiO_4, hydrous titanium oxide, or titanic acid, as it is sometimes called, appears to satisfy most of the requirements of a commercial absorber. It is relatively cheap, capable of large scale manufacture, fairly selective, durable and requires inexpensive chemicals in the associated extraction cycle (Haigh, 1974). The method of extraction consists of contacting activated titanium hydroxide granules with seawater, possibly for a period of up to 4 days to achieve maximum absorption (von Kienlin, 1976). Current procedure enables 50% of the uranium to be extracted from the seawater (Keen, 1968). Absorption is completed when the absorber is saturated, or 'loaded', and it is then washed, or eluted, with an eluant such as ammonium carbonate solution. The eluant re-dissolves the uranium and forms a weak pregnant liquor. Uranium can either be directly precipitated from this liquor by adding excess common salt - a method referred to here as the Direct Solvent Extraction (D.S.E.) process. Alternatively, salt consumption can be reduced considerably if the pregnant liquor is pre-concentrated by removing, or 'stripping', surplus ammonia with steam. This technique is called the Steam Stripping (S.S.) process (Davies et al, 1964; Harrington et al, 1974).

Apart from common technical and economic considerations, any proposed seawater treatment scheme must satisfy two basic requirements to be commercially feasible. The first is that the extraction plant must be able to handle large amounts of seawater, since intial uranium concentration is so low. To produce even a modest annual output of 500 tonnes of U_3O_8, an extraction scheme would have to achieve an average throughput of 5 million litres of seawater per second. Consequently the scheme must either have a large collection area to intercept the natural flow of seawater or it must incorporate huge high-speed water pumps. Whichever method is used, a large storage area will also be necessary, if absorption periods are long, and hence all schemes are likely to need vast capital investment.

The second requirement of any successful extraction plan is that the inflow and outflow of the plant must not be allowed to mix. To avoid the re-cycling of treated, or 'spent', seawater and thus prevent the dilution of untreated seawater, ingoing and outgoing streams must be carefully segregated. This could be achieved by a number of methods such as using the outflow to feed a desalination plant or by physically separating the two streams with some sort of barrier.

Although numerous schemes have been put forward, no pilot or commercial plants have yet been built. Economic studies of some proposals have already been conducted, however, and early results indicated total costs of $10 to $100 per pound U_3O_8 (Davies et al, 1964; Keen, 1968; Bieniewski, Persse and Brauch, 1971). More detailed analysis (Harrington et al, 1974) has revealed a minimum cost of $300 per pound U_3O_8.

These estimates may be compared with typical prices for uranium from conventional ores of between $8 and $30 per pound U_3O_8 during the early 1970's.

Energy analysis has also been performed elsewhere on two particular extraction designs (Taylor and Walford, 1974) and results indicated total energy requirements for yellow cake of between 2×10^7 and 6×10^8 MJ(t) per tonne U_3O_8. This is somewhat higher than the current energy requirement of uranium concentrate; $10^6 - 10^7$ MJ(t) per tonne U_3O_8.

In contrast to other studies, the energy requirements of various schemes operating under differing conditions were deduced here by formulating a general equation which describes all energy inputs to the extraction process. Using information on the details of operation and basic hydrodynamics, the following expression was derived;

E_s = energy input per unit mass U_3O_8 extracted from seawater $= E_A + E_B + E_C + E_D$

where,

E_A = e.r. of seawater pumping (10^6 MJ/te U_3O_8)

$$= \left[\left(\frac{18.5}{C} \times \frac{R}{K} - v_o\right)^2 + (9.8 \times H)\right] \times 0.4 + \left[\frac{F \times L}{\sqrt{A} \times C^2} \times \left(\frac{R}{K}\right)^2 \times 1743\right]$$

E_B = e.r. of elution (10^6 MJ/te U_3O_8) = e_{B1}

E_C = g.e.r. of chemical processing (10^6 MJ/te U_3O_8)

$$= (m_1 \times e_{C1}) + (m_2 \times e_{C2}) + (m_3 \times e_{C3}) + (m_4 \times e_{C4})$$

E_D = g.e.r. of plant and equipment (10^6 MJ/te U_3O_8)

$$= \left[\left(k_1 \times \frac{e_{D1}}{l_1}\right) + \left(k_2 \times \frac{e_{D2}}{l_2}\right) + \left(k_3 \times \frac{e_{D3}}{l_3}\right) + (k_4 \times e_{D4})\right] \times \frac{K}{R}$$

Note;

$\frac{1}{C}$ = total capital cost of the scheme per unit collection area ($/m^2: 1965 - 1970)

R = annual uranium production rate (tonnes U/year)

K = total capital cost of the scheme ($: 1965 - 1970)

v_o = average sea current velocity (m/s)

H = effective pumping head (m)

F = fractional coefficient of friction of the pipework

L = average length of pipework per duct (m)

A = total cross-sectional area of ducts (m^2)

e_{B1}= average e.r. of elution (10^6 MJ/te U_3O_8)

m_1 = absorber make-up rate (tonnes/tonne U_3O_8)

e_{C1}= g.e.r. of the absorber (10^6 MJ/tonne)

m_2 = eluant make-up rate (tonnes/tonne U_3O_8)

e_{C2}= g.e.r. of the eluant (10^6 MJ/tonne)

m_3 = extractant consumption rate (tonnes/tonne U_3O_8)

e_{C3}= g.e.r. of the extractant (10^6 MJ/tonne)

m_4 = steam consumption rate (tonnes/tonne U_3O_8)

e_{C4}= g.e.r. of steam (10^6 MJ/tonne)

k_1 = fraction of capital cost attributed to civil works

e_{D1}= e.i. of civil work construction (MJ/$: 1965 - 1970)

l_1 = operational lifespan of civil works (years)

k_2 = fraction of capital costs attributed to pumping machinery

e_{D2}= e.i. pumping machinery construction (MJ/$: 1965 - 1970)

l_2 = operational lifespan of pumping machinery (years)

k_3 = fraction of capital costs attributed to chemical plant

e_{D3}= e.i. of chemical plant construction (MJ/$: 1965 - 1970)

l_3 = operational lifespan of chemical plant (years)

k_4 = fraction of capital costs attributed to annual maintenance

e_{D4}= e.i. of maintenance work (MJ/$: 1965 - 1970)

The form and structure of the above equation enables the effect of various important parameters on the total energy requirement of seawater treatment schemes to be examined comprehensively. Although some parameters are obviously determined by the particular type of extraction technique and seawater handling scheme in use, others can be regarded as independent. Parameters treated as independent in this study include the energy intensities, e_{D1} to e_{D4}, and lifetime estimates, l_1 to l_3, for capital plant and equipment. Energy intensities obtained from the data base are shown in table E.1 and lifetime estimates calculated from civil engineering data (Wilson, 1965; Overman, 1968) are illustrated in table E.2.

A number of parameters are determined by the particular extraction technique used and results for the titanium hydroxide absorber cycle are given in tables E.1 and E.2. Table E.1 shows energy requirements, e_{B1}, e_{C1}, e_{C2}, e_{C3} and e_{C4}. The energy requirement of elution, e_{B1}, mainly consists of the energy input to eluant pumping and was deduced from projected operating figures (Harrington et al, 1974). The g.e.r.'s of titanium hydroxide absorber, e_{C1}, ammonia eluant feedstock, e_{C2}, common salt extractant, e_{C3}, and steam, e_{C4}, were obtained from the data base.

Chemical consumption rates, m_1 to m_4, illustrated in table

Table E.1 : Energy requirements and energy intensity estimates for inputs to the titanium hydroxide seawater extraction process.

Parameter	Units	Value	
		minimum	maximum
e_{B1} : eluant pumping e.r.	MJ/kg U_3O_8	15 (e)	460 (e)
e_{C1} : titanium hydroxide e.r.	MJ/kg	2 (e) + 0.4 (t)	16 (e) + 7 (t)
e_{C2} : ammonia e.r.	MJ/kg	0.15 (e) + 18 (t)	0.23 (e) + 26 (t)
e_{C3} : salt e.r.	MJ/kg	0.01 (e) + 0.15 (t)	0.01 (e) + 0.47 (t)
e_{C4} : steam e.r.	MJ/kg	3.5 (t)	3.5 (t)
e_{D1} : civil works e.i.	MJ/$	2.4 (e) + 52 (t)	6 (e) + 70 (t)
e_{D2} : pumping machinery e.i.	MJ/$	4.8 (e) + 49 (t)	3.6 (e) + 76 (t)
e_{D3} : chemical plant e.i.	MJ/$	3.6 (e) + 65 (t)	6 (e) + 70 (t)
e_{D4} : maintenance e.i.	MJ/$	2.4 (e) + 52 (t)	6 (e) + 70 (t)

Table E.2 : Operating parameters for the titanium hydroxide seawater extraction process.

Parameter	Units	Value
m_1 : titanium hydroxide make-up rate	te H_4TiO_4/te U_3O_8	56
m_2 : ammonia make-up rate	te NH_3/te U_3O_8	260
m_3 : salt consumption rate	te NaCl/te U_3O_8	1700 (D.S.E. route) 8.5 (S.S. route)
m_4 : steam requirement	te/te U_3O_8	- (D.S.E. route) 74000 (S.S. route)
l_1 : civil works life	years	80
l_2 : pumping machinery life	years	35
l_3 : chemical plant life	years	25

E.2 were estimated from expected operating conditions (Davies et al, 1964; Keen, 1968; Harrington et al, 1974).

The basic parameters for titanium hydroxide treatment shown in table E.1 and E.2 were used to evaluate the energy requirement terms for elution, E_B, and chemical processing, E_C. Minimum and maximum values of these terms were deduced by combining the lowest and highest estimates of the basic parameters. This enabled the full range of E_B and E_C to be calculated and results are illustrated in table E.3. Values of E_C for both the Direct Solvent Extraction (D.S.E.) technique and the Steam Stripping (S.S.) method are also given in this table. In the following analysis the full extent of the total energy input, E_s, was estimated by incorporating the minimum value of E_C for the D.S.E. route, giving $(E_s)_{min}$, and the maximum value of E_C for the S.S. route, resulting in $(E_s)_{max}$, in the general equation.

Most remaining parameters depend on the actual details of the seawater handling scheme in question. Such dependent variables include the factors, k_1 to k_4, which describe the division of total capital cost, K, between the various types of equipment installed. The term, 1/C, which is the ratio of capital cost, K, to total intake duct collection area, A, is also a dependent variable determined by design. The effective pumping head, H, and duct pipework length, L, are influenced by the characteristics of the scheme, whilst the value of the friction coefficient of the pipework, F, is affected by the type of material used. Siting of the scheme determines the sea speed, v_o.

Table E.3 : Energy requirement terms for the titanium hydroxide process.

Term	Value (10^6 MJ/tonne U_3O_8) minimum	maximum
E_B ; e.r. of elution :	0.015 (e)	0.460 (e)
E_C ; e.r. of chemical processing		
- D.S.E. route :	0.170 (e) + 5 (t)	0.970 (e) + 8 (t)
- S.S. route :	0.150 (e) + 260 (t)	0.955 (e) + 270 (t)

To investigate the energy requirement of any particular scheme it is necessary to evaluate appropriate values of the dependent variables, k_1 to k_4, v_o, H, F, L and A. Consequently the resulting terms E_A and E_D, which refer to seawater pumping and capital equipment respectively, depend on the specific scheme under examination. In contrast the terms E_B and E_C, describing extraction operations, are scheme-independent.

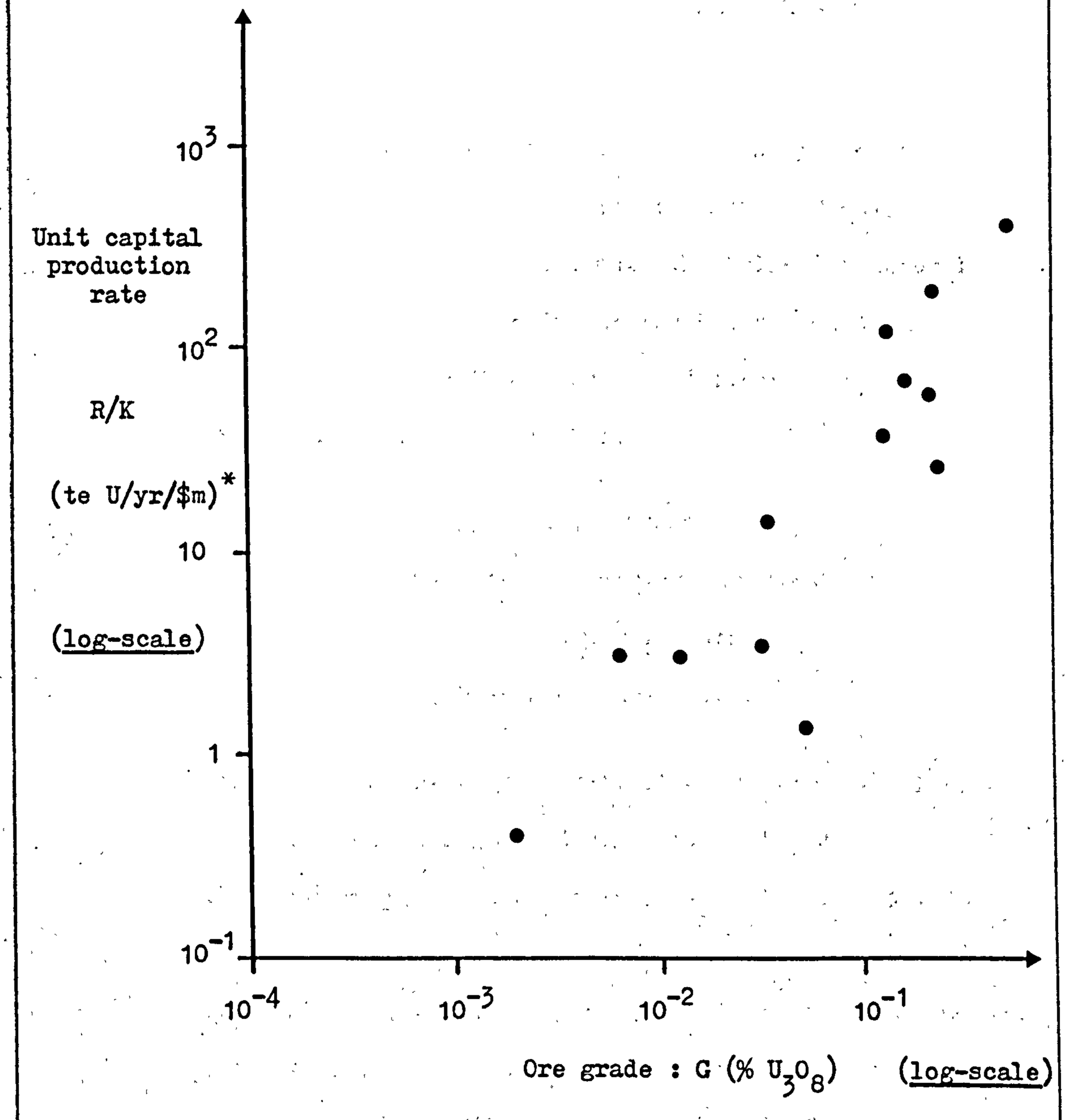
The term R/K is the fundamental variable in the general equation. Since R is the annual uranium output and K is the initial capital cost, the ratio R/K relates production to investment and subsequently indicates the rate of return on capital for any scheme. In addition to demonstrating the role of capital in production, this variable can be used to compare the unit capital cost of uranium produced by different techniques, i.e.;-

$$\text{Capital cost} = \left(\frac{K}{R}\right) \times \frac{1}{\text{Operating life in years}} \quad \$m/\text{tonne U}$$

Comparison of seawater extraction schemes and conventional ore mining and processing operations can also be achieved through the term R/K and figure E.1 shows a plot of R/K with ore grade, G, for individual mines and mills.

For seawater extraction schemes in particular, the variable R/K influences the total energy input to uranium production, E_s, through the terms E_A and E_D in the general equation. The energy requirement of water pumping, E_A, is proportional to R/K, whilst the energy requirement of plant and equipment, E_D, is inversely proportional. Hence R/K can be used to distinguish schemes that achieve similar rates of output by different means. For example,

Figure E.1 : Variation of the unit capital production rate with ore grade for conventional mines and mills.



KEY : ● = observed data point

* capital cost assessed at 1968 prices

a small scheme which relies on a high pumping speed to produce a given output rate has a large value of R/K that increases E_A and decreases E_D. Conversely, a large, capital intensive scheme which attains the same output rate has a lower value of R/K that subsequently reduces E_A and enhances E_D.

Using the general equation, the total energy input, E_s, to various schemes based on the titanium hydroxide extraction process was deduced. The following general schemes which appear to fulfil the basic operating conditions of high throughput and influx-outflux segregation were investigated;-

(1) Free-flow schemes

(2) Man-made barrier pumping scheme

(3) Overland pumping scheme

(4) Deep-sea pumping scheme

Free-flow schemes rely on natural currents or the pumped flow of water through auxilary equipment, such as a desalination plant or coastal power station, to provide a continuous supply of seawater for processing. Coastal tidal-flow schemes currently appear to be the most popular type of proposal, probably because of their relative simplicity (Keen, 1968; Harrington et al, 1974). In general, these schemes incorporate a multiple bay system filled by tidal motions to produce a steady flow of uranium-bearing seawater through the absorber beds. Consequently no large-scale pumping is required. Since natural flow rates are usually comparatively low, vast volumes of water must be processed to achieve a realistic annual uranium output. Hence coastal tidal-flow schemes

are large - a proposed plant capable of producing 1000 tonnes U_3O_8 each year in the Menai Straits off the North Wales coast would impound an area of roughly 100 square kilometres (Keen, 1968).

Specific operating parameters and subsequent energy input terms describing typical coastal tidal-flow schemes are shown in table E.4 and these were used to evaluate the following upper and lower limits to the total energy requirement, E_{s1};

$$(E_{s1})_{min} = \left[0.2 + 0.06x\frac{K}{R}\right](e) + \left[\ 5 + 1.2x\frac{K}{R}\right](t)\ 10^6 MJ/te\ U_3O_8$$

$$(E_{s1})_{max} = \left[1.4 + 0.18x\frac{K}{R}\right](e) + \left[270 + 2.1x\frac{K}{R}\right](t)\ 10^6 MJ/te\ U_3O_8$$

The variation of these limits with R/K is illustrated in figure E.2 where a conversion factor of 4 MJ(t) per MJ(e) is assumed.

For coastal tidal-flow schemes the production rate per unit capital, R/K, is determined by the local sea current speed, v_o. Consequently there is a limit to R/K fixed by the maximum achieveable sea speed, $(v_o)_{max}$. Using the hydrodynamic equation for E_A, the following relationship, giving the maximum value of R/K, was derived;

$$\left(\frac{R}{K}\right)_{max} = 3.8 \times (v_o)_{max} \quad \text{tonnes } U_3O_8/\$m/\text{year}$$

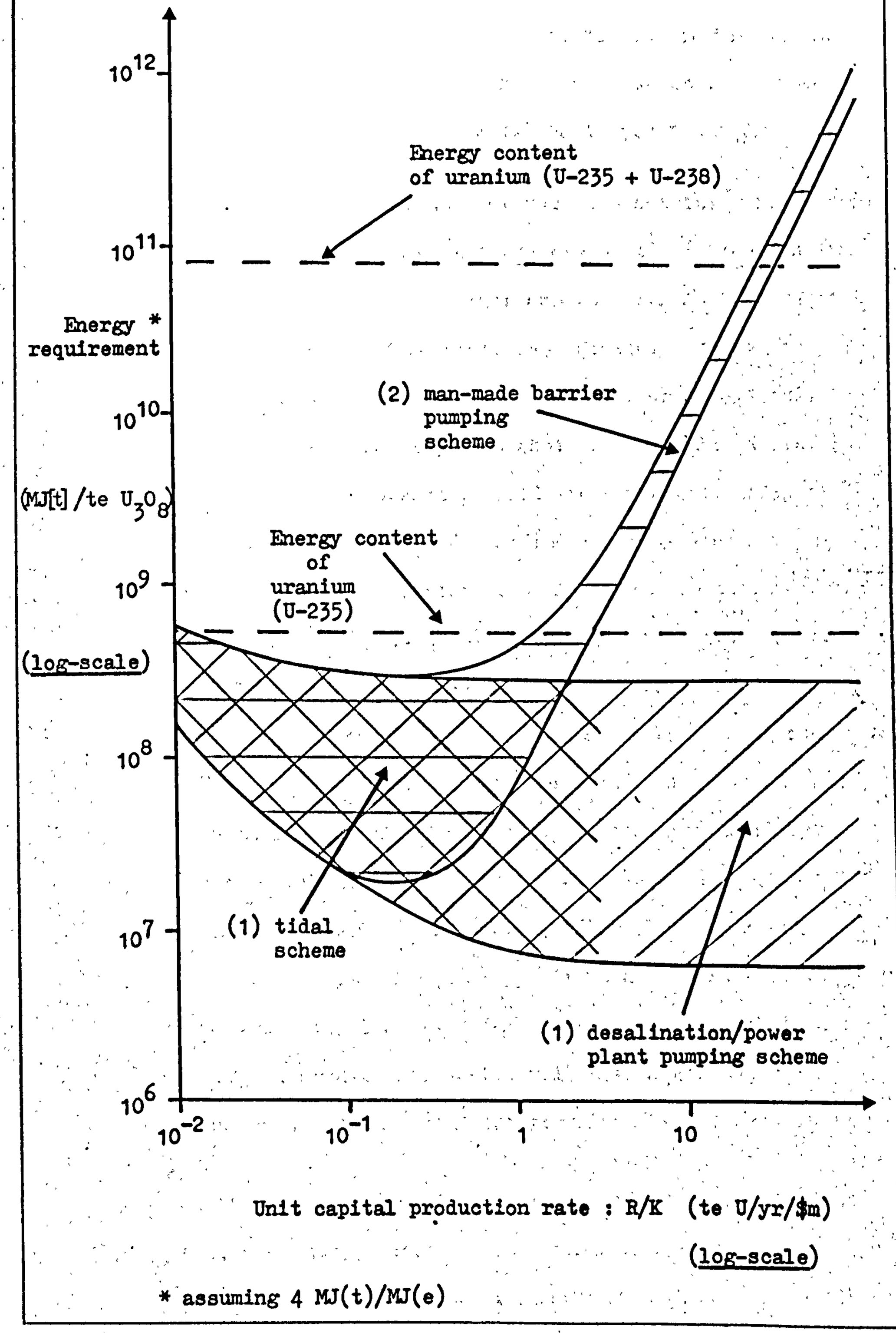
where $(v_o)_{max}$ = highest current velocity in metres/second

Since the maximum current speed is approximately 0.8 metres per second (Neumann, 1968; Pickard, 1975), the maximum possible annual extraction rate for such schemes is 3 tonnes U_3O_8 per million dollars invested. With this value of R/K the energy requirement of uranium production is

Table E.4 : Factors for natural and 'free' flow seawater uranium extraction systems (scheme 1).

Factor	Units	Value	
		minimum	maximum
E_A	: 10^6MJ/te U_3O_8 :	-	-
E_B	: 10^6MJ/te U_3O_8 :	0.015 (e)	0.460 (e)
E_C	: 10^6MJ/te U_3O_8 :	0.170 (e) + 5 (t)	0.955 (e) + 270 (t)
k_1	: -	0.20	0.35
k_2	: -	-	-
k_3	: -	0.30	0.50
k_4	: -	0.005	0.005

Figure E.2 : Variation of the energy required to produce uranium from seawater with free-flow and assisted-flow schemes.



* assuming 4 MJ(t)/MJ(e)

between 6×10^6 and 3×10^8 MJ (t) per tonne U_3O_8. This can be compared with a result of 2×10^7 MJ (t) per tonne U_3O_8 for a tidal scheme with R/K equal to 2 tonnes U_3O_8 per million dollars per year obtained by other researchers (Taylor and Walford, 1974). These values are higher than the current energy requirement of uranium produced from commercial ores of 1 to 2×10^6 MJ(t) per tonne U_3O_8.

High energy requirements, however, are not the main factors restricting the development and use of tidal-flow uranium extraction facilities. Other aspects such as high capital costs are serious disadvantages. Capital cost can be reduced by combining uranium extraction units with tidal power schemes, assuming no technical difficulties are encountered. At present, world tidal power capacity is relatively small and, if modified to process uranium as well as produce electricity, could only provide an annual output of roughly 10 tonnes U_3O_8. Although tidal power capacity could be expanded, the number of feasible sites is limited (Wilson, 1973) and it is unlikely that more than 10^4 tonnes U_3O_8 could be produced yearly in this manner. This is almost half the current world nuclear fuel demand and significantly less than the forecast demand of 3×10^5 tonnes U_3O_8 required annually by 2000 (Nuclear Energy Agency, 1975). Consequently, although such schemes may supplement local nuclear fuel supply, it seems that this method of uranium production will not make a major contribution to an expanding global nuclear power system.

One method of increasing the unit capital production rate, R/K, and subsequently avoiding natural restrictions on uranium supply is to pump seawater through the extraction

system. Since pumping is likely to be expensive, schemes which rely on the pumped flow of associated, ancillary equipment are obviously attractive. By combining an uranium extraction unit with a desalination plant (Khan, 1972), a coastal power station (Haigh, 1974) or a coastal steel mill (Japanese Times, 1975), seawater pumping can be regarded as 'free' in relation to both financial accounting and energy analysis. Consequently, the parameters given in table E.4 and the equation for E_{s1} still describe the energy inputs to such schemes ($E_A = 0$; $k_2 = 0$). Figure E.2 shows the variation of the total energy input, E_{s1}, which is constant beyond the previous limit of $(R/K)_{max}$ for tidal-flow schemes.

Hence, despite possible financial savings, combined free-flow schemes cannot achieve lower energy requirements than the most favourably sited $\left[v_o = (v_o)_{max}\right]$ tidal-flow schemes. Although capital costs can be reduced, there are a number of serious practical difficulties with such schemes. Storage is an important problem - a unit extracting 1 tonne of U_3O_8 annually would require absorption beds equivalent in volume to 20 super-tankers. The size of the combined plant can also restrict uranium production - it has been estimated that the largest commercial desalination installation currently available could only produce 1 tonne of U_3O_8 yearly and that the cooling water used by a typical coastal burner reactor power station only contains 7% of the uranium it consumes (Haigh, 1974). The operation of combined schemes also depends on the demand for products other than uranium, such as water, electricity, steel, etc. Consequently, it appears that such schemes are unlikely to be regarded as

major primary sources of nuclear fuel.

The main disadvantages of combined systems can be avoided by independent pumping schemes. These would probably be less expensive than tidal-flow systems and yet could produce realistic amounts of uranium. High-speed pumping, however, can cause problems of influx-efflux mixing which would impair the efficiency of uranium extraction. To ensure that uranium-free seawater is not re-processed, schemes which rely on pumping must provide suitable means of segregating untreated and treated water. All remaining schemes considered here offer different solutions to this particular problem.

Scheme 2, which involves pumping water through absorption beds situated in a man-made barrier that separates two significantly large stretches of sea, is a common proposal. Parameters which enable the energy requirement, E_{s2}, to be evaluated for this technique are given in table E.5. To reduce the capital cost it was assumed that the barrier is narrow and low, i.e. small values of $L/\sqrt{A}$ and H, and is made out of cheap raw materials, i.e. a low value of 1/C and high value of F. The following maximum and minimum values of the total energy input, E_{s2}, were obtained from the general equation;

for $R/K \ll v_o$; $(E_{s2})_{min} \equiv (E_{s1})_{min}$ and $(E_{s2})_{max} \equiv (E_{s1})_{max}$

for $R/K \gg v_o$;

$$(E_{s2})_{min} = \left[18\left(\frac{R}{K}\right)^2 + 0.12\left(\frac{K}{R}\right) + 0.2\right](e) + \left[1.7\left(\frac{K}{R}\right) + 5\right](t) \quad 10^6 MJ/te\ U_3O_8$$

$$(E_{s2})_{max} = \left[28\left(\frac{R}{K}\right)^2 + 0.16\left(\frac{K}{R}\right) + 9.3\right](e) + \left[2.2\left(\frac{K}{R}\right) + 270\right](t)\ 10^6 MJ/te\ U_3O_8$$

Table E.5 : Factors for the man-made barrier seawater pumping uranium extraction system (scheme 2).

Factor		Units	Value	
			minimum	maximum
$\frac{1}{C}$	:	$\$/m^2$	0.36	0.45
$-v_0$	:	m/s	0.80	0
H	:	m	0	2
F	:	-	0.003	0.009
$\frac{L}{\sqrt{A}}$	:	-	10^{-3}	10^{-2}
E_B	:	10^6 MJ/te U_3O_8	0.015 (e)	0.460 (e)
E_C	:	10^6 MJ/te U_3O_8	0.170 (e) + 5 (t)	0.955 (e) + 270 (t)
k_1	:	-	0.2	
k_2	:	-	0.4	
k_3	:	-	0.3	
k_4	:	-	0.005	

Figure E.2 shows the variation of these limits which rise rapidly for increasingly higher values of the unit capital production rate, R/K. Consequently, schemes which give a high rate of return on capital investment use quite substantial amounts of energy to produce uranium. These results may be compared with the theoretical amount of heat available from the complete fission of uranium - 5×10^8 MJ(t) per tonne U_3O_8 for U-235 fission and 7×10^{10} MJ(t) per tonne U_3O_8 for U-235 fission, U-238 conversion and subsequent Pu-239 fission (see figure E.2).

High fuel consumption and the lack of suitable sites may restrict the use of this type of scheme. The construction of a long, narrow barrier for influx-efflux segregation can be expensive and difficult. This can be avoided by using a natural barrier to separate the incoming and outgoing water. Hence scheme (3), which involves pumping seawater overland, may seem an interesting solution. To investigate the effect of the pumping distance, L, on the energy input, E_{s3}, to this sort of proposal, the general equation was re-written in terms of L. Table E.6 shows the parameters used to obtain the following average energy requirement;

$$(E_{s3})_{average} =$$

$$\left[(50L^2 + 2L^3) \times \left(\frac{R}{K}\right)^2 \times 10^{-4} + 0.1\left(\frac{K}{R}\right) + 5\right](e) + \left[2\left(\frac{K}{R}\right) + 140\right](t)$$

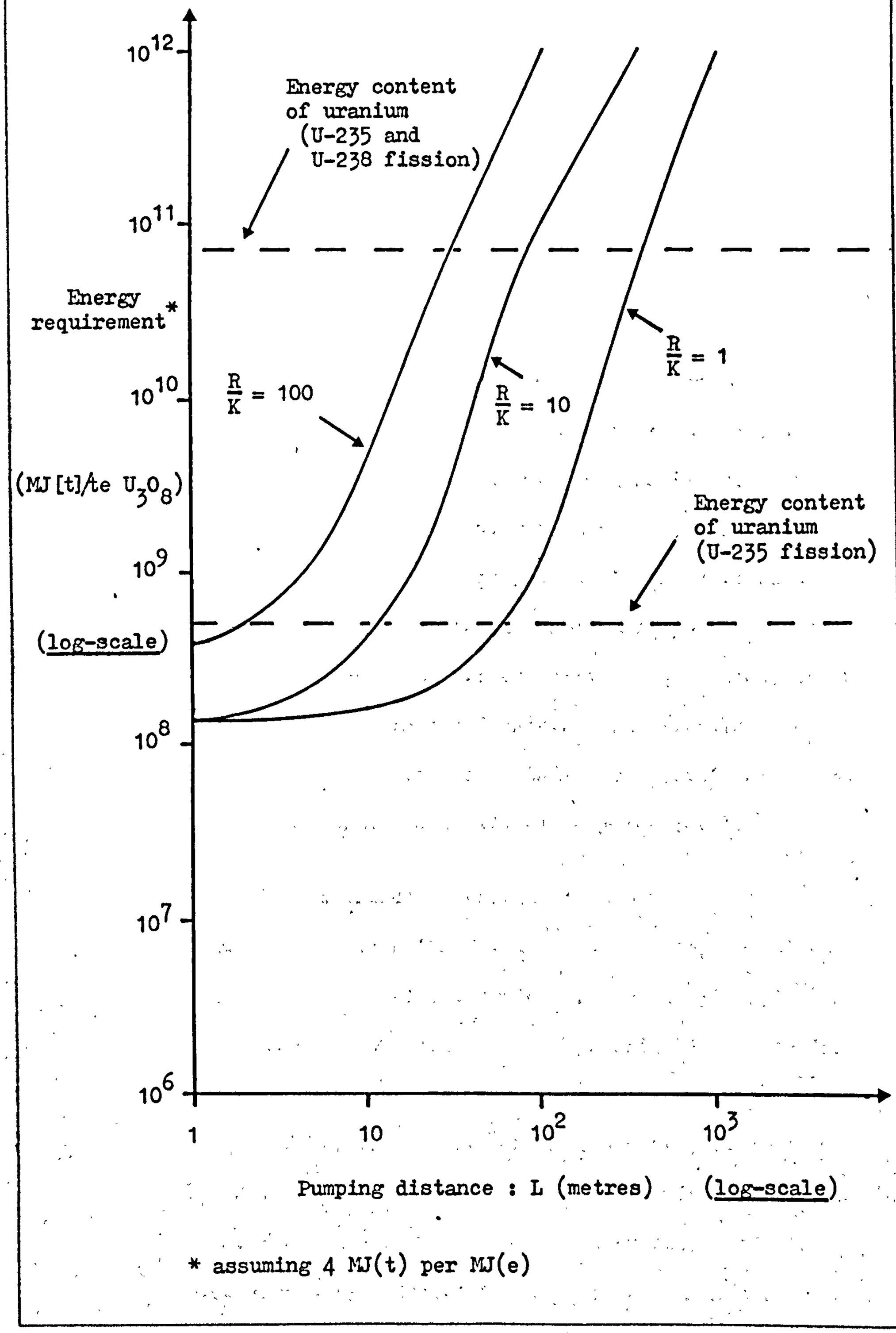
10^6 MJ/te U_3O_8

The variation of E_{s3} with L for fixed values of R/K is illustrated in figure E.3. This indicates that the total energy required by an overland pumping scheme is quite sensitive to the pumping distance, L. By comparing these

Table E.6 : Average factors for an overland seawater pumping uranium extraction system (scheme 3).

Factor		Unit	Value
$\frac{1}{C \times L}$	:	$/m^3	6
v_o	:	m/s	0
H	:	m	5
F	:	-	0.0025
A	:	m^2	1
E_B	:	10^6 MJ/te U_3O_8	0.24 (e)
E_C	:	10^6 MJ/te U_3O_8	0.56 (e) + 140 (t)
k_1	:	-	0.2
k_2	:	-	0.4
k_3	:	-	0.3
k_4	:	-	0.005

Figure E.3 : <u>Effect of pumping distance on the energy required to produce uranium from seawater</u> (scheme 3).



* assuming 4 MJ(t) per MJ(e)

results with the theoretical amount of energy available from uranium (see figure E.3), it can be seen that schemes giving currently commercial returns on capital (R/K > 10 te U_3O_8/yr/$m) are impractical, in energy terms, for pumping distances greater than a few tens of metres. Consequently the seemingly attractive idea of an uranium extraction plant straddling the Panama isthmus (which conveniently separates two vast oceans) is unlikely to be successful, financially or 'energetically', since a pumping distance of about twenty thousand metres is involved.

The final method considered here is the deep-sea pumping technique, scheme (4), which consists of a floating pumping and uranium processing installation with feed pipes that can extract water from the ocean depths. Such schemes may be regarded as practical propositions when surface currents promote the rapid mixing of treated and untreated seawater and where barrier segregation, of one form or another, is not feasible. Additionally such schemes could be used to exploit the majority of oceanic uranium resources which lie at very low levels.

Although the world's oceans contain 5×10^9 tonnes of uranium, less than 1% of this is in the top 30 metres amenable to coastal extraction schemes considered previously. The total amount of uranium available for these schemes is not limited to resources of 3×10^7 tonnes of uranium in the surface, however, since this layer is being continually replenished with uranium-bearing water from rivers and ascending ocean currents. These processes only restrict the <u>rate</u> at which uranium can be

taken from the surface layer. The maximum global extraction rate is determined by river flow, currently amounting to about 3×10^4 tonnes of uranium annually (Davies et al, 1964), and by the vertical mixing time, or number of years required for water from a given level to reach the surface.

Mixing times vary according to depth and, although periods of half a year have been observed for layers only a few metres below the surface, mixing times of a thousand years have been estimated for depths over five thousand metres (Broeker, 1974; King, 1975). Mixing times reflect the rate of exchange between lower levels and the surface and consequently they can be used to determine maximum extraction rates. From basic oceanographic information (Strahler, 1963) the limiting rate for the majority (95%) of oceanic resources, which occupy depths of 5000 metres or less, was estimated to be between 10^6 and 10^8 tonnes of uranium per year.

Although this production rate is somewhat higher than the expected annual demand of about 3×10^5 tonnes of uranium in 2000 (Nuclear Energy Agency, 1975), technical factors may reduce this in practice. To achieve higher global production rates from the oceans it appears that surface extraction schemes would be impractical and deep-sea pumping installations would be necessary. The amount of energy, E_{s4}, required to produce uranium by such methods was investigated by re-writing the general equation in terms of the pumping depth or head, H (also putting L = H). Typical parameters for scheme (4) are shown in table E.7 and these were used to obtain the following expression

Table E.7 : Average factors for a deep-sea pumping uranium extraction system (scheme 4).

Factor		Units	Value
$\frac{1}{C \times H}$	:	$\$m/m^3$	0.01
v_o	:	m/s	0
F	:	-	0.003
$\sqrt{A}$	:	m	1
L	:	m	H
E_B	:	10^6 MJ/te U_3O_8	0.24 (e)
E_C	:	10^6 MJ/te U_3O_8	0.56 (e) + 140 (t)
k_1	:	-	0.2
k_2	:	-	0.4
k_3	:	-	0.3
k_4	:	-	0.005

for the average energy requirement;

$$(E_{s4})_{average} =$$

$$\left[4xH + 0.014x\left(\frac{R}{K}\right)xH^2 + 0.00045x\left(\frac{R}{K}\right)^2xH^3 + 0.1\left(\frac{K}{R}\right) + 0.8\right](e)$$

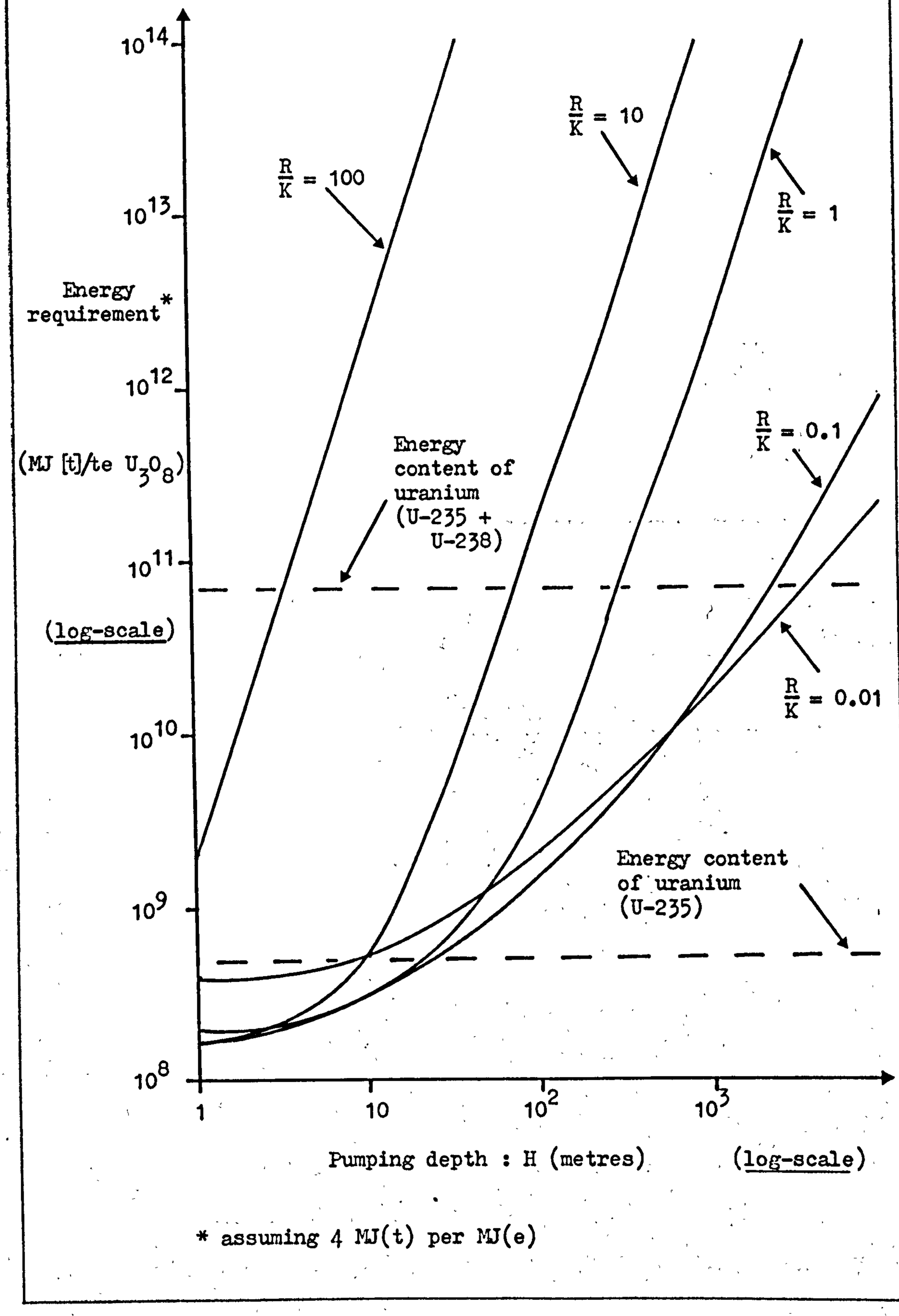
$$+ \left[2x\left(\frac{K}{R}\right) + 140\right](t) \qquad 10^6 MJ/te\ U_3O_8$$

Figure E.4 shows the variation of this relationship, which is quite sensitive to the pumping head, for various values of R/K. This figure demonstrates that for realistic values of R/K (i.e. R/K $\geqslant$ 10 tonnes uranium per year per $m) the pumping depth must not exceed a few tens of metres if the energy required to produce a given quantity of uranium is to be less than the theoretical amount of energy it can release by fission. Hence, with these sort of energy and capital-return criteria, it would appear that such schemes are not viable for exploiting deep-lying oceanic uranium resources.

To conclude this analysis, results of all the various extraction schemes examined are compared with the energy requirements of conventional ore mining and processing. The average energy requirements of uranium production by these methods are illustrated in figure E.5. Data points for individual mines and mills were obtained from figure E.1 and other empirical information sources.

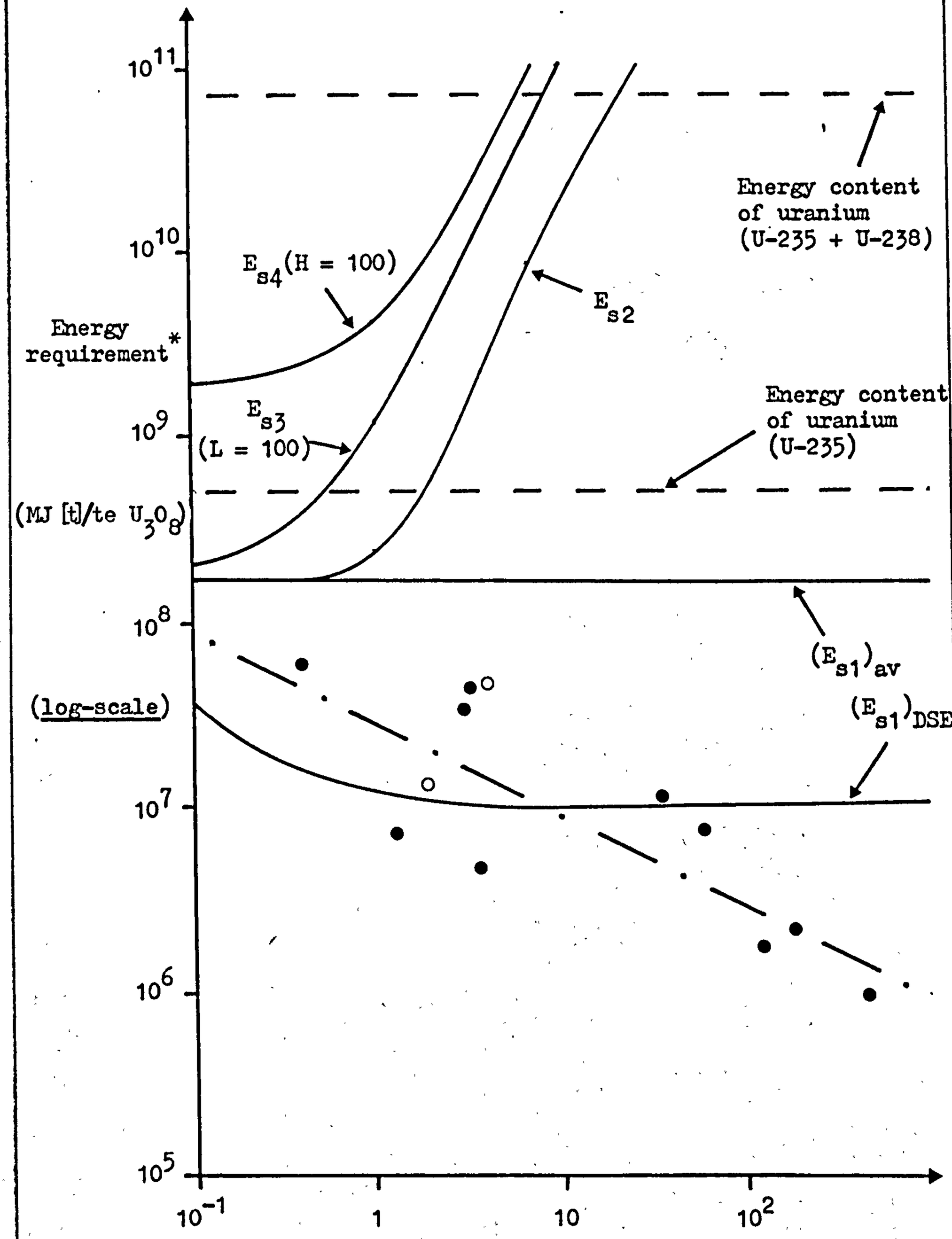
Figure E.5 shows that the extraction of uranium from seawater only becomes comparable, in energy terms, with conventional methods of production for very low values of R/K. This implies that, on average, seawater processing schemes would only begin to be competitive when uranium is obtained from low-grade ores, probably less than 10 ppm

Figure E.4 : <u>Effect of pumping depth on the energy required to produce uranium from seawater</u> (scheme 4).



* assuming 4 MJ(t) per MJ(e)

Figure E.5 : Energy requirements of uranium production by conventional ore mining and processing and seawater extraction schemes.



* assuming 4 MJ(t) per MJ(e)

● = mine and mill data ; O = Taylor and Walford, 1974

U_3O_8 (see figure E.1). As regards the theoretical energy available from uranium, man-made barrier, overland and deep-sea pumping techniques (schemes 2, 3 and 4) are all 'unprofitable' for high values of R/K and only marginally 'economic' for low values. Of all the schemes considered, tidal-flow and combined desalination operations (scheme 1) seem to be the most attractive. Using the energy-saving Direct Solvent Extraction process the average energy input to these schemes is reduced to about 10^7 MJ(t) per tonne U_3O_8 which is similar to the energy requirement of uranium from ores with grades less than 0.05% U_3O_8.

Appendix F : Uranium resources

A resource is a natural source of materials which supports a given need and the total amount of a particular resource available at any one time according to specified conditions is generally called the resource base. Many factors can determine the specific effort required to extract useful materials from resources and the variation of the amount of a resource with such factors is often referred to as a resource profile. Evaluation of resource profiles can play a fundamental role in the investigation of industrial growth and economic development. Estimation of the uranium resource profile, described here, is an important part of the study of nuclear power development.

Most previous assessments of uranium resources have attempted to relate resources to the current estimated cost of their production (eg. Cambel, 1965; Boxer, Haussermann, Cameron and Roberts, 1971; Bowie, 1974, World Energy Conference, 1974; Nuclear Energy Agency, 1975). In general, such studies have indicated the amount of recoverable uranium in certain cost categories, usually expressed in US dollars per pound triuranium octoxide ($/lb U_3O_8). Although presentation of the uranium resource profile in terms of cost is obviously necessary for the study of nuclear power economics, this approach has important limitations.

Calculation of the current costs of exploration, and ore mining and processing is usually based on numerous implicit assumptions, changes in which fundamentally alter the shape of the resource profile. Technological

improvements can reduce costs, whilst inflation can increase them. Such changes can also affect different operations differently. The cost of uranium produced by an old mine which employs a large work force, for example, is more vulnerable to rising wage rates than the cost of uranium from a newer, mechanised pit. Consequently traditional resource estimates based on current production costs are sensitive to changes which are almost impossible to predict and therefore such results cannot be freely used in an unqualified manner to forecast the future effect of resources on nuclear power growth.

The most fundamental disadvantage of these results is that estimated production costs implicitly incorporate current fuel price data. This approach cannot be used when discussing the future of a fuel supply industry such as the nuclear power system, since the price of uranium concentrate influences the cost of nuclear electricity and the price of electricity affects the cost of uranium concentrate. If all fuel used by all industries was generated by nuclear reactor power stations there would be an important economic feedback mechanism running back through the nuclear fuel cycle which would strongly influence the relative cost of resources.

At the moment fuel costs are only a minor component of the unit cost of uranium concentrate, generally less than 20%, and nuclear power hardly supplies any of the fuel used in most mines and mills. Hence the feedback effect is weak and generally not apparent. This leads to the common belief, usually fostered by traditional resource

analysis, that every type of resource has a cost and that, provided the price is right, even rocks which contain minute amounts of uranium are viable sources of nuclear fuel.

The presence of a potentially strong feedback effect contradicts this idea which implies a practically infinite, though expensive, uranium resource base. If the use of nuclear power were to increase and the quality, or ore grade, of commercially mined ore falls, then both the amount and cost of energy used to extract uranium from resources will rise. Hence the energy requirement and cost of nuclear electricity increases rapidly as ore grades decrease. Eventually, when the amount of energy required to manufacture nuclear material equals the total amount of fuel it can generate, the entire system becomes uneconomic and costs reach infinity. Such reasoning suggests a finite and absolute limit to the quality and quantity of uranium resources that can be used in nuclear power systems.

One purpose of energy analysis is to identify this limit. This can be achieved by describing the uranium resource profile in terms of the basic geological and technical characteristics which determine the cost and, in particular, the energy requirement of producing uranium from resources. To calculate accurately the amount of energy required to produce uranium from any particular ore deposit, it would be necessary to completely specify all features of the deposit. Such detailed information is

not generally available for all deposits and in practice analysis must be confined to the most influential characteristics.

From initial investigation it appears that the most important factors which determine the energy requirements of uranium production are the ore grade, rock characteristics, the waste to ore ratio and ore mining and processing techniques. In this analysis ore grade, G, a commonly quoted parameter which indicates the amount of triuranium octoxide in the ore, is regarded as the fundamental variable. Whilst ore grades can vary over a wide range of values (from approximately one part per million upwards), other factors are less variable. Consequently rock composition, deposit characteristics and production methods superimpose fluctuations on the main variation of energy requirement with ore grade and are generally treated as secondary variables.

Estimates of the uranium resource profile, in terms of ore grade, were deduced from research papers, geological reports, resource assessments, etc., and results are shown in table F.1. Resource totals are divided into convenient, though arbitrarily defined ore grade categories and a number of different profiles are illustrated.

Column (1) shows the most recent estimate of resources commonly referred to as 'reasonably assured resources' (World Energy Conference, 1974; Nuclear Energy Agency, 1975). Reasonably assured resources are defined as the 'uranium which occurs in known ore deposits of such grade, quantity and configuration that it could be recovered

Table F.1 : Estimated uranium resource profiles.

Average ore grade (U_3O_8)	(1) Reasonably assured resources	(2) Historical U.S. orthodox ore total	(3) Current world total	(4) Extrapolated world total
	(tonnes recoverable U_3O_8)			
0.5% - 2.0% :	1.5×10^5	-	1.5×10^5	1.5×10^5
0.2% - 0.5% :	4.7×10^5	6.9×10^5	8.8×10^5	8.7×10^6
0.1% - 0.2% :	8.7×10^5	1.7×10^6	2.2×10^6	2.1×10^7
0.05% - 0.1% :	4.7×10^5	1.4×10^5	7.8×10^5	1.8×10^6
0.02% - 0.05% :	3.8×10^5	-	4.5×10^5	4.5×10^5
0.01% - 0.02% :	-	$(6.8 \times 10^5)^*$	6.8×10^5	4.5×10^6
50 ppm - 0.01% :	-	$(4.9 \times 10^6)^*$	4.9×10^6	4.9×10^6
20 ppm - 50 ppm :	-	$(2.2 \times 10^9)^*$	2.2×10^9	3.2×10^9
10 ppm - 20 ppm :	-	$(7.9 \times 10^6)^*$	7.9×10^6	9.9×10^7
5 ppm - 10 ppm :	-	$(3.5 \times 10^6)^*$	1.7×10^{11}	1.7×10^{11}
1 ppm - 5 ppm :	-	-	5.6×10^{10}	5.6×10^{10}
10 ppb - 1 ppm :	-	-	5.2×10^8	5.2×10^8
1 ppb - 10 ppb :	-	-	2.5×10^9	2.5×10^9

* unorthodox ores including phosphates, shales and granites.

ppm = parts per million ; ppb = parts per billion

within a given production cost range, with currently proven mining and processing technology' (Nuclear Energy Agency, 1975). The resources shown in column (1) indicate the amount of uranium currently available for extraction at a cost of less than $30 per pound U_3O_8 (1st. January 1975). Included in this estimate are 1.3×10^6 tonnes U_3O_8 considered as reserves which could be produced for less than $15 per pound U_3O_8. In common with columns (3) and (4), column (1) gives world resources which exclude uranium from the USSR, Eastern Europe and China.

The historial orthodox ore resource base of the USA is shown in column (2). This indicates the total amount of uranium available prior to large scale mining in the USA, based on production as well as resource data (Cambel, 1965; U. S. Atomic Energy Commission, 1972; Woodmansee, 1972; Youngberg, 1973; World Energy Conference, 1974; Nuclear Energy Agency, 1975). The total shown in table F.1 represents the sum of reasonably assured resources, estimated additional resources and other inferred, unappraised or undiscovered resources. Estimated additional resources are defined as the 'uranium surmised to occur in unexplored extensions of known deposits or in undiscovered deposits in known uranium districts, and which is expected to be discoverable and could be produced in a given cost range' (Nuclear Energy Agency, 1975).

A compilation of current world resource estimates which covers all categories of uranium deposit is given in column (3). In addition to standard information on orthodox resources (International Atomic Energy Agency,

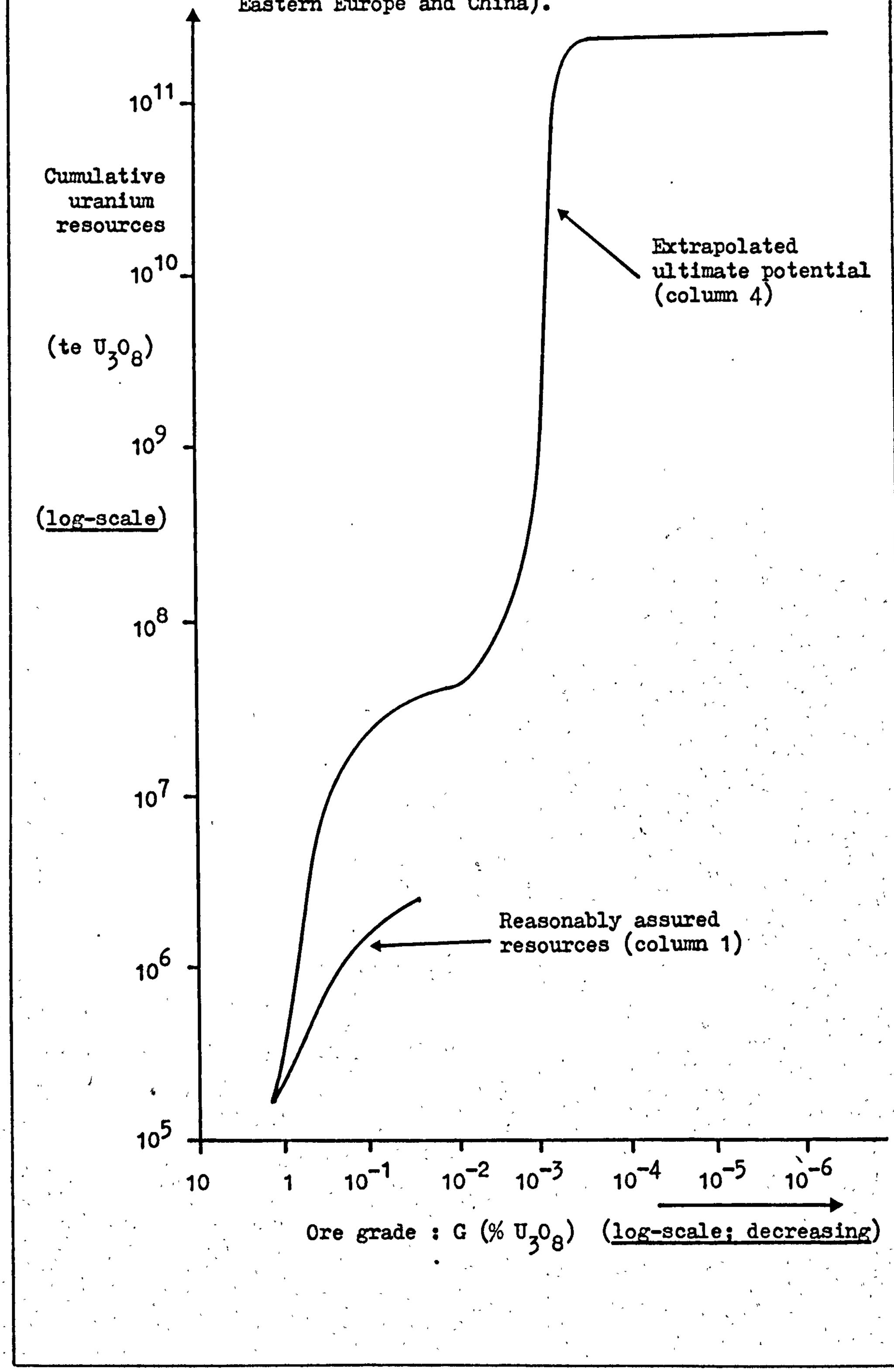
1970b;World Energy Conference, 1974; Nuclear Energy Agency, 1975), this includes latest data on recent ore finds, assessments of unorthodox resources and estimates of the amount of uranium which occurs in common rocks and seawater. The data on new ore discoveries were based on geological survey results published in journals such as Mining Magazine, Nuclear Engineering International, etc., and figures for unconventional resources were obtained from research papers (Hurst, Crouse, Brown and Ross, 1966; Bieniewski, Persse and Brauch, 1971). Estimates of the uranium contained in crustal abundance were deduced from geochemical data (Fairbridge, 1972; Rösler and Lange, 1972) and lithological information (Strahler, 1963; Menard, 1974). Approximate values of the recoverable crustal uranium were evaluated assuming that practical operations could be expected to extract 100% of the uranium in common rocks covering about 20% of world land area to a currently mineable depth of roughly 0.3 kilometres. Figures of recoverable oceanic resources were calculated from a value of uranium concentration equal to 3.3 ppb (Wilson et al, 1960), oceanographic data (Strahler, 1963) and an assumed extraction efficiency of 50% (Keen, 1968).

Column (4), which shows extrapolated world resources, represents a postulated estimate of the total amount of recoverable uranium. These results were based on current established resource information. The main assumption used is that the USA, as one of the most thoroughly explored regions of the earth, can be regarded as a very crude resource model for the rest of the world (cf. Cambel, 1965). Consequently an order of magnitude

estimate of uranium potential at high grades can be found by increasing the US orthodox ore resource profile (column 2) by the ratio of world land area (excluding the USSR, Eastern Europe and China) to US land area (i.e. x 12.6). In addition, resources in grade categories which have no equivalent in column (2) were simply represented by the resources in column (3). World resources of unorthodox source of uranium were derived from separate information. The amount of uranium contained in phosphatic minerals was deduced by multiplying total world, non-US, phosphate rock reserves of 2.5×10^{10} tonnes (Slack, 1966) by an assumed average uranium ore grade of 150 ppm U_3O_8 and adding the result to similar US phosphatic resources of 6.8×10^5 tonnes U_3O_8 (Bieniewski et al, 1971). Uraniferous shale resources were calculated in a similar manner using an assumed average ore grade of 35 ppm U_3O_8 and oil shale reserve information (Gustafson, 1969; Hubbert, 1969). Values for crustal and oceanic uranium were obtained from column (3).

In common with most other resource assessments these results are subject to some degree of uncertainty. Although some results, especially those of column (4), may be regarded as rather crude compared to the conclusions of other, more sophisticated geological analyses (see, for example, Chapman, 1976a), such estimates should, at least, give some impression of the potential scope of uranium resources. This can be further demonstrated by figure F.1 which shows the variation of cumulative resources with ore

Figure F.1 : <u>Variation of cumulative recoverable uranium resources with ore grade</u> (excluding resources of the USSR, Eastern Europe and China).



grade. The lower bound represents reasonably assured resources (column 1), whereas the upper consists of extrapolated world resources (column 4). The figure indicates that there is a wide difference between a minimum amount of 2.3×10^6 tonnes of U_3O_8 currently available and a possible ultimate potential of 2.3×10^{11} tonnes U_3O_8.

Appendix G : Refining and conversion of uranium concentrate

After it has been extracted from ore, uranium concentrate must be refined and converted into material suitable for the production of nuclear fuel. Raw uranium concentrate received at the refinery generally contains upwards of 80% triuranium octoxide, U_3O_8. Certain important impurities such as the neutron absorber, boron, are often present and must be removed from uranium to obtain reactor-grade nuclear material. Refining processes which achieve this occupy the initial stages in the refinery plant. The remaining function of the refinery consists of converting the purified uranium into suitable compounds for further steps in the fuel cycle. The type of conversion processes used and the form of the final product obtained depend on the particular sort of nuclear fuel required.

Reactor type determines the physical and chemical characteristics of nuclear fuel manufactured from uranium concentrates. The MAGNOX reactor uses uranium metal, clad in a magnesium alloy known as magnox, as a fuel. The fission physics of this reactor are based on natural uranium, or uranium containing the typical isotopic abundance of 0.71% uranium-235. Consequently no enrichment, or enhancing of the U-235 nuclei, is required and refineries which supply such reactors produce purified uranium metal ingots. The CANDU reactor also consumes natural uranium fuel, but in the form of pelletised uranium dioxide, UO_2. This material is converted from uranium compounds obtained from the early stages of refining.

All other major reactor types which include the Advanced Gas-cooled Reactor (AGR), the Steam Generating Heavy Water Reactor (SGHWR), the Boiling Water Reactor (BWR) and the Pressurised Water Reactor (PWR) use enriched uranium dioxide fuel which contains an artificially increased proportion of the fissile isotope U-235. Most common enrichment techniques (see appendix H) utilise the gaseous uranium compound uranium hexafluoride, UF_6, or 'hex'. Hence refineries which produce material for such reactors include processes for the conversion of refined uranium compounds to hex.

Energy requirements of refining and conversion of uranium for various reactor fuel cycles were evaluated using information from standard chemical engineering textbooks, industrial flowcharts and research papers. Chemistry handbooks (Shreve, 1967; Eister and Kennedy, 1974) and analytical reports (US Environmental Protection Agency, 1973; Pigford, Keaton and Mann, 1973; Organisation for Economic Co-operation and Development, 1974; U.S. Bureau of Mines, 1975) were particularly useful sources of data. Energy requirements for numerous processes and products were deduced by combining figures on costs and fuel and chemical consumption with estimates from the data base (see chapter 3). Results are shown in tables G.1, G.2, G.3 and G.4.

Table G.1 indicates the amount of energy required to refine raw uranium concentrate. Refining of impure concentrate, or 'yellow cake', which usually contains a mixture of ammonium diuranate, magnesium diuranate and sodium uranate, consists of dissolution in nitric acid followed by solvent extraction. Very pure uranyl nitrate, $UO_2(NO_3)_2 6H_2O$, is

obtained by the selective extraction of uranium with organic reagents such as ethyl ether or tributyl phosphate (TBP) diluted in kerosene or hexane (Shreve, 1967). Uranium dioxide, UO_2, or black oxide powder, is recovered from refined uranyl nitrate solution by denitration.

Table G.2 summarises the energy requirement of denitration. The removal of nitrate and the production of uranium dioxide can be accomplished by a number of different procedures. At present, the most widely used method consists of heating uranyl nitrate to form uranium trioxide, UO_3, or orange oxide (Shreve, 1967), which is reduced to uranium dioxide in a continuous fluidised bed reactor furnace with hydrogen produced from dissociated ammonia (Mattson, 1967). Alternatively, denitration can be achieved by direct flame reduction using propane (Hedley, Roehrs and Trask, 1964) or by reacting uranium trioxide with carbon dioxide to form uranyl carbonate, UO_2CO_3, which is then reduced to uranium dioxide by hydrogen in the presence of metal catalysts and organic promoters (Warren and Forward, 1961). Uranyl nitrate can also be converted to uranium dioxide by the Ammonium DiUranate (ADU) process. This technique which is used in conjunction with yellow cake containing a high proportion of sodium (Mattson, 1967) consists of washing the uranyl nitrate with ammonium sulphate to produce sodium-free ammonium diuranate, $(NH_4)_2U_2O_7$. After filtering and drying uranium dioxide is obtained by hydrogen reduction (Shreve, 1967).

Uranium dioxide powder can be used to produce fuel pellets for CANDU reactors or converted to uranium metal for MAGNOX reactors. Additionally, it can be used as a feed

material for the manufacture of uranium hexafluoride gas. This conversion involves reacting uranium dioxide with hot hydrogen fluoride gas in a hydrofluorination furnace to obtain the intermediary product, uranium tetrafluoride, UF_4, or green salt (Shreve, 1967). This is then reacted with very pure, dry fluorine gas, produced electrolytically from hydrofluoric acid, in a fluorination reactor (Mattson, 1967). Uranium hexafluoride gas which is formed is cooled and bottled into cylinders. The energy requirement of these processes is illustrated in table G.3.

Numerous methods can be used to produce uranium metal in the refinery and various energy requirements are shown in table G.4. The fused-fluoride electrolysis route starts with uranium dioxide which is incorporated into a consumable graphite anode. This is used in the electrolysis of a hot electrolyte containing barium difluoride, lithium fluoride and re-cycled scrap uranium tetrafluoride. Very pure uranium metal is produced which is tapped directly from the molten electrolyte (Piper and Leifield, 1962). One of the most common techniques of producing reactor-grade metal uses uranium tetrafluoride as a starting material. This is mixed with magnesium and rapidly heated to a very high temperature in a thermit-type bomb furnace to obtain molten uranium metal and magnesium slag (Mattson, 1967; Shreve, 1967). An alternative route begins with uranium hexafluoride gas which is directly reduced with sodium metal. Uranium metal and sodium fluoride slag are separated by dissolution in water (Scott, 1963).

Table G.1 : Energy input to the refining of uranium concentrate and production of uranyl nitrate.

Process input (per te uranium output)	Energy requirement (MJ/te U) minimum	maximum
DISSOLUTION ;		
- electricity = 8900 kWhe	: 32000(e)	32000(e)
- nitric acid = 710-960 kg	: 64(e) + 7500(t)	130(e) + 13000(t)
- calcium oxide = 280-380 kg	: 20(e) + 2100(t)	27(e) + 3500(t)
SOLVENT EXTRACTION ;		
- sodium carbonate = 50 kg	: 17(e) + 940(t)	23(e) + 1800(t)
- sodium hydroxide = 18 kg	: 96(e) + 340(t)	120(e) + 350(t)
- sulphuric acid = 20 kg	: 50(t)	250(t)
- organic chemicals = 20 kg	: 1400(t)	6500(t)
- water = 4×10^5 kg	: 3000(t)	3600(t)
Total capital = $110-210	: 70(e) + 7300(t)	1330(e) + 15000(t)
TOTAL (rounded)	: 32300(e) + 22600(t)	33600(e) + 44000(t)

Table G.2 : Energy inputs to the production of uranium dioxide from uranyl nitrate.

Process input (per te uranium output)	Energy requirement (MJ/te U) minimum	maximum
DENITRATION ;		
- natural gas = 45 - 2400 m^3	: 1800(t)	96000(t)
- water = 4 x 10^5 kg	: 3000(t)	3600(t)
Total capital = $50 - 110	: 30(e) + 3700(t)	670(e) + 8400(t)
TOTAL	: 30(e) + 8500(t)	670(e) + 108000(t)

Table G.3 : Energy inputs to the conversion of uranium dioxide to uranium hexafluoride.

Process input (per te uranium output)	Energy requirement (MJ/te U) minimum	maximum
HYDROFLUORINATION ;		
- hydrogen fluoride = 360 kg:	1200(e) + 12000(t)	2000(e) + 24000(t)
- natural gas = 2400 m^3 :	95000(t)	96000(t)
FLUORINATION ;		
- hydrofluoric acid = 170 kg:	580(e) + 5900(t)	910(e) + 11000(t)
- electricity = 2700 kWhe :	9300(e)	10000(e)
- water = 1.5 x 10^5 kg :	1200(t)	1300(t)
Total capital = $110 - 220:	60(e) + 7200(t)	1300(e) + 15000(t)
TOTAL (rounded) :	11100(e) + 121000(t)	14200(e) + 147000(t)

Table G.4 : Energy inputs to the production of uranium metal.

Process	Energy requirement (MJ/te U) minimum	maximum
Electrolytic conversion of uranium dioxide to uranium metal	28000(e) + 31000(t)	36000(e) + 53000(t)
Magnesium reduction of uranium tetrafluoride to uranium metal	5000(e) + 28000(t)	10000(e) + 43000(t)
Sodium reduction of uranium hexafluoride to uranium metal	14000(e) + 1000(t)	14000(e) + 3000(t)

Appendix H : Uranium isotope enrichment

Uranium consists of two important isotopes; uranium-235 (U-235) and uranium-238 (U-238). The natural abundance, by mass, of these isotopes is 0.71% U-235 and 99.28% U-238. The isotope U-235 is fissile which means that it can take part in energy-releasing fission reactions with neutrons. In contrast, the isotope U-238 is fertile and can only produce fissile isotopes after conversion by neutrons. Since fission reactions provide the basic source of energy in burner reactors, the amount of U-235 present in nuclear fuel determines the effective operation of such systems. Although some reactors such as MAGNOX and CANDU designs operate on natural uranium which contains 0.71% U-235, others such as the AGR, SGHWR, BWR and PWR systems use fuel which contains an increased proportion of U-235.

Consequently uranium isotope enrichment, which involves enhancing the relative abundance of U-235 in uranium, is an important part of some nuclear fuel cycles.

Isotope enrichment processes are based on slight physical dissimilarities caused by mass differences between isotopes. For low mass elements the relative difference in mass between isotopes is large and discrimination can be achieved via gross characteristics such as differences in boiling points, etc. With heavy elements such as uranium, however, the relative isotopic mass differences are small and hence the isotopes of such elements cannot be distinguished on the basis of simple gross properties. Instead, sensitive techniques must be used to separate the isotopes of uranium.

Uranium can be enriched by numerous methods. The most widespread commercial technique currently used is the gas diffusion process which is based on differences in the rates of diffusion of isotopically-different gaseous uranium compounds. Although simple in principle, the process requires vast capital investment and consumes large amounts of fuel. Moreover, the process is relatively inefficient, consuming ten million times the theoretical minimum amount of energy needed to completely separate U-235 from U-238 (McGeoch, 1977), and since fuel costs are high, techniques which consume less energy are obviously attractive.

The gas centrifuge process which is currently being developed commercially separates isotopes by means of differences in centrifugal forces acting on gas molecules in a spinning container. The technique uses roughly one tenth of the energy required by the gas diffusion process (McGeoch, 1977), but needs high-performance machinery made from special materials which can withstand very fast rotational speeds exceeding fifty thousand revolutions per minute (Barnaby, 1969; Brooks, 1970; Fishlock, 1974).

These engineering problems can be avoided by 'stationary walled centrifuge' processes such as the jet nozzle and vortex tube techniques. In these methods the gas is rotated aerodynamically inside fixed containers and specially-designed components separate the lighter isotopic gas from the natural mixture. Although each unit of these types of centrifuge achieves a higher degree of isotope separation than a similarly-sized diffusion unit,

early development work has indicated that such techniques may consume two and a half times as much energy as conventional enrichment methods (Avery and Kehoe, 1970).

Much faster rotational velocities and subsequently higher levels of isotope separation than possible with mechanical centrifuges can be accomplished with the plasma centrifuge process. Initial research with rotating, partially and fully ionised hot gases, or plasmas, suggests that energy requirements for enrichment will be lower than those of many other methods (Nuclear Engineering International, 1975a).

Research into the possibility of using laser light to separate isotopes is currently receiving much attention. Numerous techniques are being examined but all rely on much the same principle - that different isotopes absorb slightly different wavelengths of light. Laser light, which is highly-monochromatic radiation, is used because isotopic differences of absorption wavelengths are very small. The particular isotope that has been selectively excited by the exactly correct wavelength of laser light can be removed from other, non-excited isotopic species by means of photoionisation, photodissociation or photochemistry. In the photoionisation method excited atoms become electrically charged and are separated from neutral atoms by electromagnetic fields.
Photodissociation relies on the eventual splitting of excited molecules and the photochemistry technique is based on the fact that molecules excited to high vibrational levels are chemically more reactive than

unexcited molecules. It has been estimated that processes using laser light may be up to one thousand times more efficient than the diffusion enrichment method (McGeoch, 1977).

The total amount of energy required by these commercial and experimental enrichment techniques was evaluated using data from textbooks, technical articles, economic reports, industrial flowsheets and research papers. The results of this analysis are shown in tables H.1 and H.2. Table H.1 indicates the fuel consumption of different processes and table H.2 illustrates the estimated indirect energy requirements for equipment,maintenance, ancillary supplies, etc.

The results shown in tables H.1 and H.2 are presented in two different types of enrichment units. Energy requirements of the gas diffusion, gas centrifuge, jet nozzle, vortex tube and plasma centrifuge processes are given in terms of mass units of separative work (eg. tonnes of Separative Work Units; te SWU). Such processes can achieve variable levels of isotope enrichment and the separative work unit is a function of initial and final U-235 enrichment, and the percentage of U-235 remaining in rejected waste, or tails. Laser separation techniques can usually produce only one level of enrichment, between 60% and 100% U-235, and differently enriched products are obtained by mixing this material with natural uranium containing 0.71% U-235. The energy required to obtain a desired enrichment level of p_p% U-235 from material enriched to p_e% U-235 by these laser processes can be

evaluated, and compared with other results, by multiplying the values shown in tables H.1 and H.2 by the following factor;

$$\frac{p_p - 0.71}{p_e - 0.71}$$

where,

p_e = initial % U-235 enrichment level achieved

p_p = final % U-235 enrichment level desired

The direct energy requirements, unit capital costs and total costs of producing one tonne of 3.1% U-235 enriched uranium by various methods may be compared by referring to table H.3. This is a typical level of enrichment for PWR reactor fuels and a standard concentration of 0.25% U-235 in the enrichment wastes, or tails, was assumed.

Table H.3 shows that, although the widely used gas diffusion process can enrich uranium at a favourable cost compared to other methods, its fuel consumption is quite high. This arises because the uranium hexafluoride gas used in the process must be passed through many separating membranes to obtain suitable enrichment levels. High pressures must be maintained in the membrane-compressor units, or cascades, and the gas must be pumped through many kilometres of pipework during treatment. Fuel supply is so important that some enrichment plants are linked to their own power stations (eg. Muller, 1974; Owen, 1975). Since present-day energy requirements are twice the theoretical minimum (Valéry, 1971), there appears to be little scope for improvement with this method compared with the reductions other processes can achieve. Consequently

new techniques which offer much lower theoretical and practical fuel consumption, and hence reduced costs, attract much interest.

The gas centrifuge process is one of the newer techniques that may eventually supplant the gas diffusion method. Uranium hexaflouride gas is the basic feedstock for this process which uses rapidly rotating centrifuges to separate U-235 from U-238. Gas which contains a higher proportion of U-235 is called the light fraction and this is collected near the axis of the centrifuge. Many centrifuge units in series are required to obtain the desired level of enrichment but fewer cascades are needed than the diffusion technique. Despite this, costs are high because the centrifuge machinery is expensive (Mohrhauer, 1972; Roberts, 1973). Costs are expected to fall, however, with commercialisation (Avery, Bogaardt, Jelinek-Fink and Parry, 1971).

Stationary-walled centrifuge processes such as the jet nozzle and vortex tube methods operate on similar principles to the gas centrifuge technique. Both use a mixture of uranium hexafluoride and hydrogen or helium gas and it is hoped that final commercial costs will compete effectively with present costs. The jet nozzle process invented in West Germany uses a more concentrated gas mixture and lower pressures than the vortex tube technique currently being developed by the South African Atomic Energy Commission and the Uranium Enrichment Corporation of South Africa. At the moment energy requirements of both processes are quite high (see tables H.1 and H.3),

although the theoretical minimum energy required is roughly a tenth of current experimental values (Rippon, 1975).

Plasma centrifuge systems can apparently achieve relatively low fuel consumption, although final costs for these laboratory devices cannot yet be determined. The two versions of the plasma centrifuge process presently being studied use partially ionised uranium hexafluoride gas and fully ionised uranium metal or uranium dioxide vapour.

Many laser separation techniques are being examined since these processes offer substantial advantages over conventional enrichment methods. Particularly prominent are the Garching radial plasma isotope separation process, or laser Q-device (Nuclear Engineering International, 1975a), and the Lawrence Livermore Laboratories laser two-step photoionisation process (Ozaki et al, 1976). More development work is required on such techniques, however, before they can be fully commercialised.

Numerous other processes have been investigated in the past and are being researched at the moment. Separation of charged isotopes by electric and magnetic fields in devices known as calutrons was used to produce bomb-grade material in the early 1940's (Avery and Kehoe, 1970). However, such devices can only achieve small throughputs and the method is not economically competitive with current processes. A new technique has been developed by the French Atomic Energy Commission which is claimed to

produce reactor-grade enriched material using less energy and less sophisticated technology than the gas diffusion route (Mortimer, 1977).

Although improved techniques are continually being invented, it is unlikely that these will have a marked impact on the supply of enriched uranium during the next decade. Gas diffusion technology will still provide most of the enriched uranium over this period and a significant proportion of the new plant built after this time is likely to use centrifuge processes. Figure H.1 shows a forecast of world (excluding the USSR, Eastern Europe and China) enrichment requirements and planned capacity (Roberts, 1973; Nuclear Energy Agency, 1975). By combining this information with the results of tables H.1 and H.2 the average net energy requirement of enrichment during the period 1970 to 2000 can be deduced. This variation is illustrated in figure H.2. It was assumed that all new capacity installed after 1990 incorporates low fuel consumption methods such as the gas centrifuge, improved jet nozzle and vortex tube, plasma centrifuge and laser separation processes. The figure indicates that, although new processes may use considerably less energy than conventional techniques, the <u>average</u> energy requirement of enrichment is unlikely to fall to a level lower than 35% of current values before the turn of the century.

Table H.1 : Direct energy requirements of uranium isotope enrichment processes.

Process	Unit	Direct energy requirement (10^6 MJe/unit)
Gas diffusion	te SWU	9.85 ± 1.38
Gas centrifuge	te SWU	0.97 ± 0.11
Jet nozzle	te SWU	14.80 ± 2.92
Vortex tube	te SWU	12.24 ± 0.36
Fully-ionised plasma centrifuge	te SWU	0.63 ± 0.20
Partially-ionised plasma centrifuge	te SWU	1.03 ± 0.19
Laser Q-device	te 100% U235	~67
Laser two-step photoionisation	te 60% U235	1100 ± 500

Table H.2 : Indirect energy requirements of uranium isotope enrichment processes.

Process	Units	Indirect energy requirement (10^6MJ/unit)	
		electrical	thermal
Gas diffusion	: te SWU :	0.02 ± 0.01	0.29 ± 0.16
Gas centrifuge	: te SWU :	0.05 ± 0.04	0.49 ± 0.32
Jet nozzle	: te SWU :		0.18 ± 0.09

Table H.3 : Comparison of enrichment processes producing 3.1% U-235 enriched uranium with 0.25% U-235 tails.

Process	D.e.r (10^6MJe/te U)	Capital cost* ($/kg U : 1965-70)	Total cost†
Gas diffusion :	39	45	126
Gas centrifuge :	4	68	126
Jet nozzle :	59	31	174
Vortex tube :	50	n.a.	n.a.
Ionised plasma centrifuge :	3	n.a.	n.a.
Laser Q-device :	2	n.a.	63
Laser two-step photoionisation :	45	55	195

n.a. not available

* including charges, interest, etc.

† assuming $0.006/kWhe for power.

Figure H.1: World enrichment demand and capacity (excluding the USSR, Eastern Europe and China).

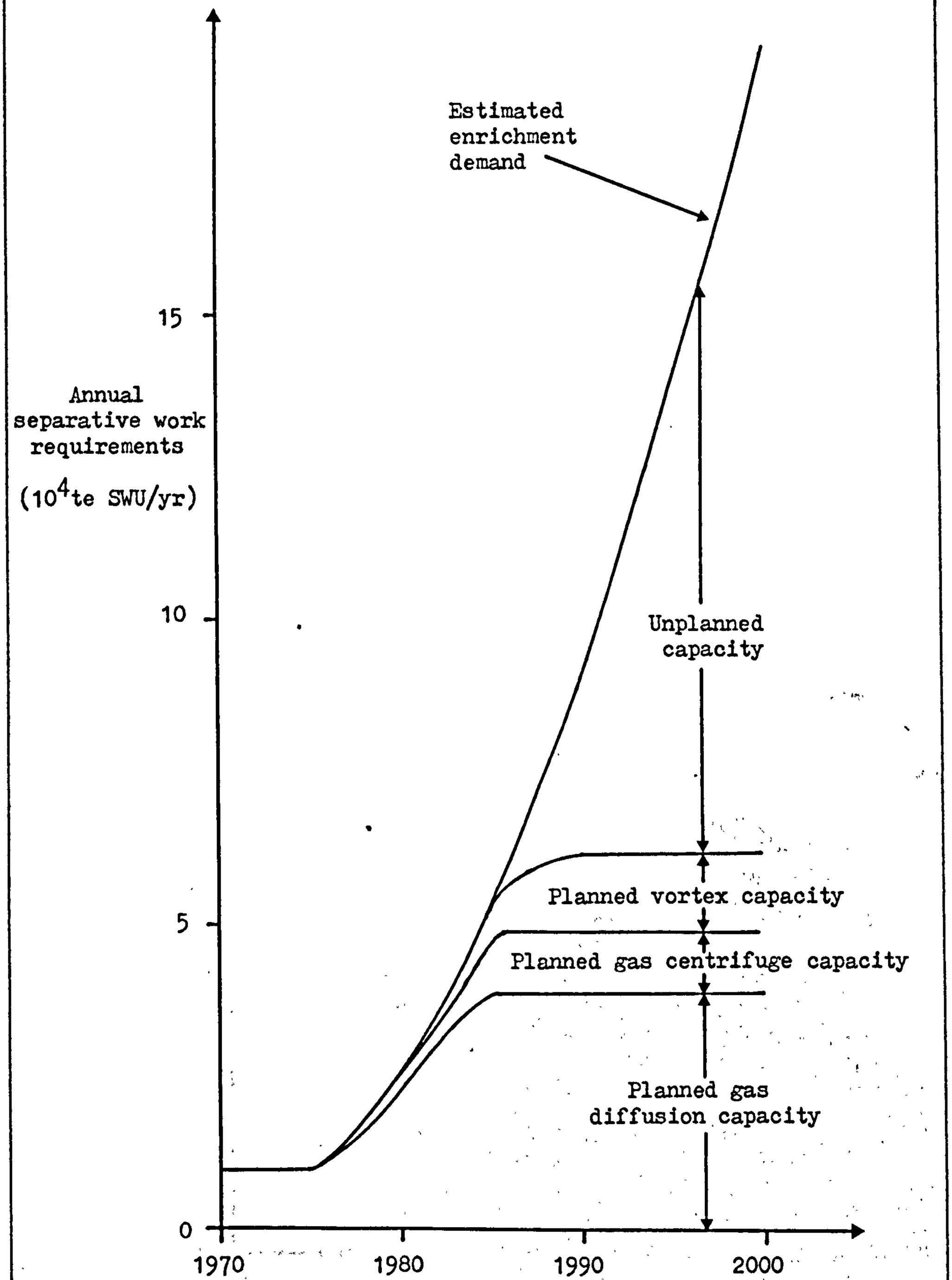
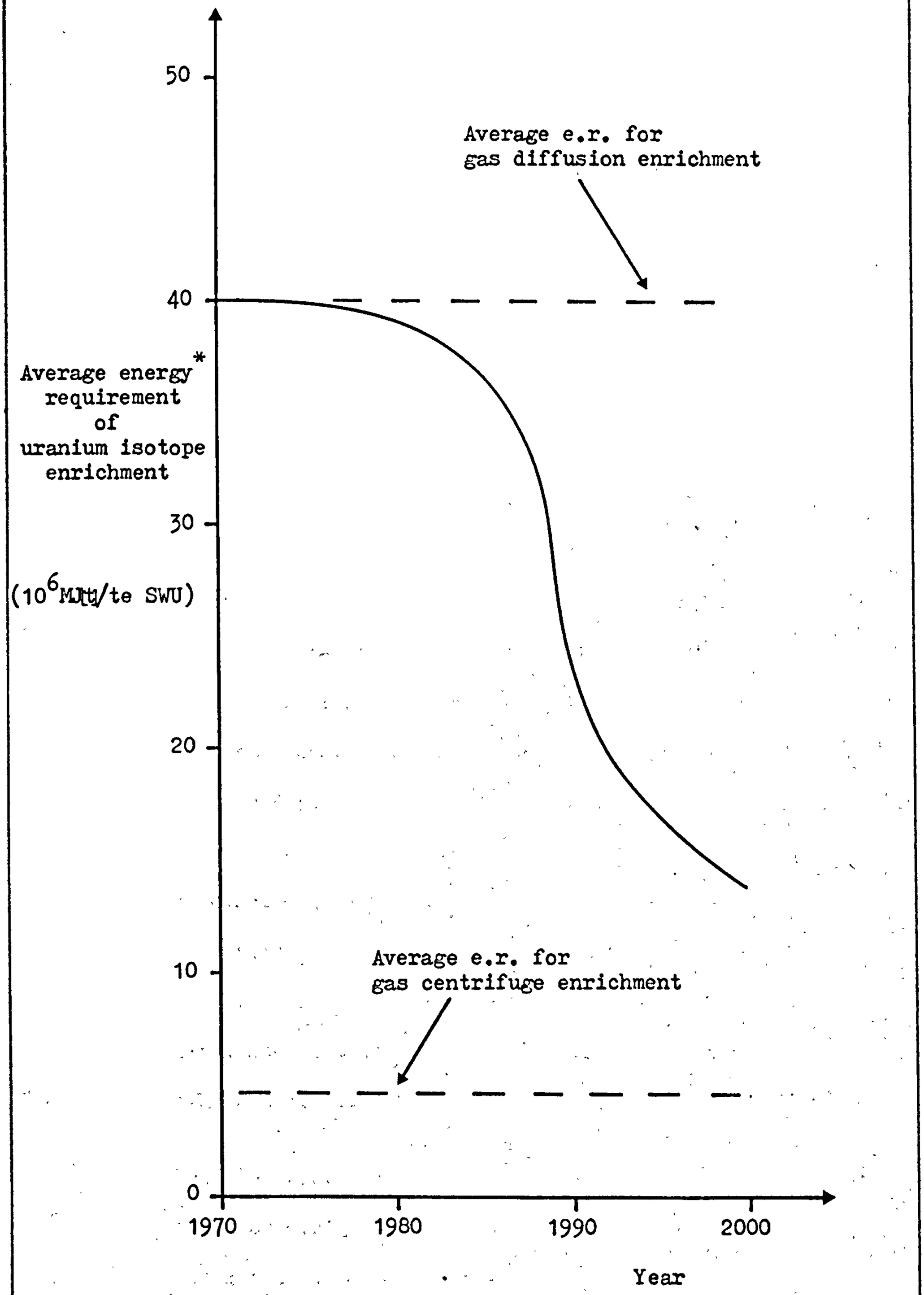


Figure H.2: Estimated average energy requirement of uranium isotope enrichment.



* assuming an electrical conversion factor of 4MJ(t)/MJ(e).

Appendix I : Fuel element manufacture

The final stage in the nuclear fuel cycle prior to placing uranium in the reactor core is fuel element manufacture. This consists of processing the nuclear material into a suitable physical and chemical form and then incorporating it into fuel slugs or pellets. These are made into fuel rods which are surrounded by special cladding materials that protect the fuel from corrosion and oxidation and also provide containment for any products formed whilst the rod is in the reactor core. Finally, fuel rods are assembled into arrays or fuel elements which are shipped to the reactor for use.

The total amount of energy required during fuel element manufacture was calculated with data obtained from textbooks (Mattson, 1967; Shreve, 1967; Eister and Kennedy, 1974), environmental flowsheets (Pigford et al, 1973; U.S. Environmental Protection Agency, 1973), reactor manuals (International Atomic Energy Agency, 1967, 1970a, 1972; Nuclear Engineering International, 1972) and research papers (Warren and Forward, 1961). This information was combined with estimates from the data base (see chapter 3), in particular those referring to the production of cladding materials such as magnesium alloys, stainless steel and zirconium alloys. Results are summarised in tables I.1, I.2 and I.3.

Details of fuel element manufacture, cladding specification, fuel rod design, etc., depend on the type of reactor being supplied. The MAGNOX system uses natural uranium metal clad in a magnesium alloy. Primary uranium metal ingots from the refinery are directly cast and machined into rods

which are then surrounded by a sheath of magnox (Mattson, 1967). CANDU reactors use natural uranium dioxide clad in a zirconium alloy. Uranium dioxide powder obtained from the refinery is formed into pellets and placed in fuel rods made of zircalloy-4 (Warren and Forward, 1961). The energy requirements for both uranium metal and uranium dioxide fuel production are given in tables I.1 and I.3.

Advanced Gas-cooled Reactor, Steam Generating Heavy Water Reactor, Boiling Water Reactor and Pressurised Water Reactor designs use enriched uranium dioxide fuel. Hence fuel fabrication plants supplying such reactors start with uranium hexafluoride gas from the enrichment plant. The gas is first vaporised with steam, hydrolysed with water and mixed with ammonium hydroxide solution to form ammonium diuranate (ADU). This precipitate is concentrated and filtered with centrifuges and dried with steam in a nitrogen atmosphere. The ammonium diuranate is then converted to ceramic uranium dioxide by calcination in a reducing ammonia atmosphere (Mattson, 1967; Shreve, 1967; U.S. Environmental Protection Agency, 1973). After sintering and grinding, uranium dioxide powder is incorporated into fuel elements. Fuel for the AGR is clad in stainless steel, that for the SGHWR and BWR in zircalloy-2 and that for the PWR in zircalloy-4. Energy requirements for uranium hexafluoride - uranium dioxide conversion are illustrated in table I.2 and the energy inputs to cladding operations are shown in table I.3.

Table I.1 : Estimated energy requirements for the fabrication of natural uranium metal fuel slugs and natural uranium dioxide fuel pellets.

Input (per te U)	Energy requirement (MJ/te U) minimum	maximum
Electricity = 4 x 10^4kWhe	: 143000(e)	143000(e)
Natural gas = 795-1100 m^3	: 31000(t)	43000(t)
Capital = $1400 - 1600	: 1000(e) + 67000(t)	10000(e) + 114000(t)
TOTAL	: 144000(e) + 98000(t)	153000(e) + 157000(t)

Table I.2 : Estimated energy requirements for the conversion of uranium hexafluoride gas to uranium dioxide fuel pellets.

Input (per te U)	Energy requirement (MJ/te U) minimum	maximum
Electricity = 5 x 10^4 kWhe	: 175000(e)	175000(e)
Natural gas = 9300 m^3	: 115000(t)	115000(t)
Water = 6.8 x 10^5 kg	: 6000(t)	6000(t)
Ammonia = 475 kg	: 8000(t)	13000(t)
Calcium oxide = 1500 kg	: 11000(t)	14000(t)
Capital = $1900	: 2000(e) + 94000(t)	11000(e) + 113000(t)
TOTAL	: 177000(e) + 234000(t)	186000(e) + 281000(t)

Table I.3 : Estimated energy requirement for producing fuel element cladding.

Fuel type	Cladding requirement (kg/te U)	Energy requirement (MJ/te U) minimum	maximum
MAGNOX ; magnesium alloy	30	1000(e) + 4000(t)	1000(e) + 6000(t)
AGR ; stainless steel	170	2000(e) + 4000(t)	4000(e) + 8000(t)
SGHWR ; zircalloy-2	125	23000(e) + 26000(t)	37000(e) + 49000(t)
CANDU ; zircalloy-4	147	27000(e) + 31000(t)	43000(e) + 57000(t)
BWR ; zircalloy-2	193	36000(e) + 40000(t)	57000(e) + 75000(t)
PWR ; zircalloy-4	204	38000(e) + 43000(t)	60000(e) + 80000(t)

Appendix J : Nuclear fuel reprocessing

Nuclear fuel containing material such as uranium produces heat in the reactor core by fission reactions between neutrons and fissile isotopes like uranium-235 (U-235). Nuclei of intermediate mass called fission products are created by fission reactions and, in addition, fissile material such as plutonium can be produced from conversion reactions involving neutrons and the fertile isotope uranium-238 (U-238). Fission products can impair the efficiency of the fuel by absorbing neutrons and causing mechanical and thermal defects. Consequently, after a given period of time, fuel which has been exposed to neutron bombardment, or irradiation, is removed from the reactor for reprocessing. This consists of separating useful fissile material such as uranium and plutonium from the fission products.

Irradiated, or 'spent', fuel elements removed from the reactor are first cooled, usually under water, to allow the residual heat to fall and short-lived fission products such as Neptunium-239 to decay. The spent fuel is then transported to the reprocessing plant in specially-cooled, crash-proof, lead-lined containers. Although new reprocessing capacity has been planned (Chartres, 1976; Clark, 1976; Rippon, 1976; Vielvoye, 1976), only a few commercial facilities are currently in operation. Hence, much spent fuel must be shipped over very large distances for reprocessing (Wright, 1975; Martin, 1976; Owen, 1976).

On arrival at the reprocessing plant the fuel elements are stored for a while under water and then prepared for

treatment by removing the cladding. This operation, known as de-canning or de-jacketing, can be achieved chemically or, more generally, by mechanical means (Flagg, 1961). This cladding, which is radioactively contaminated, is taken away for storage and the remaining fuel rods are cut into convenient lengths for dissolution in nitric acid. The resulting nitrate solution is filtered to remove any traces of insoluble cladding material and then treated by one of a number of solvent extraction processes for the recovery and purification of uranium and plutonium (Flagg, 1961).

Although numerous reprocessing methods are available, the Purex technique appears to be the most popular (Rippon, 1976). In this method uranium and plutonium nitrates are separated from fission products at the first solvent extraction stage or cycle by tributyl phosphate (TBP) dissolved in kerosene. Fission products are removed as an aqueous solution, or raffinate, and decontaminated nitrate solution is pumped to the next stage for the separation of uranium and plutonium. At this point acidified ferrous sulfamate is added to extract plutonium from the organic TBP solution. Two separate streams are formed which can be treated to obtain purified uranium and plutonium nitrate. Pure uranium nitrate is produced by repeated nitric acid recovery and TBP extraction stripping cycles which remove traces of plutonium and fission products. Plutonium nitrate is purified by successive acidification with nitric acid, oxidation with sodium nitrite and solvent extraction with TBP, followed by washing in kerosene and evaporation (Shreve, 1967).

The amount of energy required to produce uranium and plutonium nitrate from spent nuclear fuel was evaluated from data presented in the relevant literature. Although information on this part of the fuel cycle is usually difficult to obtain, operating figures were deduced from certain textbooks (Flagg, 1961; Shreve, 1967; Stevenson, Mason and Gresky, 1970; Eister and Kennedy, 1974), industrial articles (Guccione, 1964; Nuclear Engineering International, 1975b), environmental and economic analyses (Pigford, Keaton and Mann, 1973; Organisation for Economic Co-operation and Development, 1974) and research papers (Taylor and Walford, 1974; Kistemaker, 1975). Results of the analysis are summarised in table J.1 which shows energy requirements per unit mass fissile-bearing material reprocessed, that is, per tonne uranium or plutonium recovered from spent fuel.

Table J.1 : Estimated energy requirements of spent fuel reprocessing for the removal of fission products and the recovery of uranium and plutonium nitrate.

Input (per te fissile-bearing material)	Energy requirement (MJ/te fissile-bearing material)	
	minimum	maximum
Electricity = 12000 - 22000 kWhe	: 45000(e)	80000(e)
Nitric acid = 500 kg	: 5000(t)	7000(t)
Transport = 0 - 50000 km	:	30000(t)
Capital = $1400 - 2600	: 2000(e) + 82000(t)	10000(e) + 162000(t)
TOTAL	: 47000(e) + 87000(t)	90000(e) + 199000(t)

Appendix K : Nuclear waste management

Many different types of waste products can be created in the nuclear fuel cycle. The most important are those which contain radioactive materials such as reprocessing wastes. These are composed of fission products as well as traces of plutonium and uranium. In general, wastes can be divided into three categories; high - , intermediate - and low - level wastes.

High-level wastes include aqueous solutions of fission products such as americium, caesium, curium, neptunium, strontium, etc., called first cycle raffinates, from the early stages of reprocessing, and cladding from spent fuel rods. Intermediate-level wastes consist of rejected solutions from later steps in reprocessing known as second and third cycle raffinates which contain fewer fission products than first cycle raffinates. Low-level wastes are composed of liquid effluents, which only contain very small amounts of fission products, and slightly contaminated materials such as packaging.

Numerous schemes have been proposed for the use or disposal of nuclear wastes obtained from reprocessing. Energy requirements of these schemes were estimated by analysing various proposals (Schneider, 1970; Dillon, Perona and Blomeke, 1971; Clelland, 1972; Rubin, 1972). Because many schemes have not yet been used in practice it was only possible to deduce order-of-magnitude energy inputs and approximate results for some of the most promising processes are summarised in tables K.1 and K.2.

Although high-level wastes contain certain isotopes that could be recovered and used in small quantities in some industrial and research applications (Stevenson, Mason and Gresky, 1970), such effluents are regarded as hazardous materials that must be removed, either temporarily or permanently, from the immediate human environment. First cycle raffinates from reprocessing form the majority of high-level wastes. Despite high recovery efficiencies in reprocessing operations of up to 99.5% (U.S. Environmental Protection Agency, 1973), these effluents can contain as much as 0.5% radio-toxic plutonium by mass of all radionuclides present. Hence such wastes must be handled with great care.

Although the relative proportion of fission product radionuclides from spent fuel can be small, between 0.3% and 3.5% of the initial mass (Atom, 1976), the actual volume of high-level wastes is large, amounting to about ten thousand litres per tonne of fuel reprocessed (Blomeke, 1967). This arises because the concentration of fissile-bearing solutions treated in reprocessing plants is generally kept very low. Consequently the first step in the treatment of such wastes is concentration by evaporation to reduce the liquid volume by a factor of approximately one hundred (Hawkes, 1975). Following this the waste is placed in cooled storage tanks to await final disposal.

Liquid high-level wastes could, in fact, be stored permanently in such tanks until the radioactivity of the most long-lived radionuclide had decayed to about one

thousandth of its initial value. The wastes might then be considered harmless and released into the environment. However, containment for periods up to a quarter of a million years would be required and, since the waste is corrosive, a continuous supply of new storage tanks would be needed over this time. Using data on current tank design and performance characteristics (Clelland, 1972; Hawkes, 1975) and assuming that the volume of the waste remains constant, storage in self-cooled tanks would contribute about 5×10^6 MJ(t) to the energy requirements of reprocessing each tonne of fissile-bearing material. If the waste was artificially cooled through the containment period by electric pumps rated at 5 watts per litre and the waste volume was repeatedly reduced in relation to its radioactivity, the energy requirement would be approximately 10^8 MJ(t) per tonne of reprocessed fuel. These rough estimates may be compared with a theoretical energy content for the initial fuel of between 5×10^8 MJ(t) and 7×10^{10} MJ(t) per tonne.

Tank storage of high-level wastes is generally regarded as a temporary measure. Other, more permanent schemes are currently being researched and these include processes for the solidification of high-level liquid wastes by glassification (Clelland, 1972; Wright, 1976a) or by calcination (Schneider, 1970; Dillon et al, 1971). The estimated energy inputs to such schemes are illustrated in table K.1. The proposals consist of concentrating the waste by evaporation followed by interim storage in tanks for about 1 to 5 years (Hawkes, 1975; Kistemaker, 1975). The liquid would then be solidified into stable material

such as glass or calcined powder and encased in stainless steel containers designed to promote natural cooling. Many suggestions have been put forward for the disposal of these canisters, ranging from using them as low grade sources of heat in greenhouses, etc., to launching them into space by rocket (Kenward, 1974). However, the proposals which are being studied quite closely at the moment involve placing the waste in geologically stable structures such as disused salt mines (Schneider, 1970), granite and clay formations (Wright, 1976b) and in or on marine sediments (Grimwood and Webb, 1976). The energy input of such operations would mainly consist of transport since provision of disposal sites may be regarded as free, in energy terms. The energy input of this transport element varies over a wide range depending on the distances involved, but it is much less than the estimated 10^7 MJ(t) required for the proposed rocket disposal plan.

In addition to liquid effluents from reprocessing, high-level wastes also include decladding residues. The form of these wastes depends on the type of decladding procedures used - most European techniques are mechanical and these leave solid wastes, whilst acid dissolution is sometimes used in the U.S.A. which produces about ten thousand litres of liquid effluent. Solid wastes are currently stored in concrete containers or under water and liquids are stored underground in carbon steel tanks (Blomeke, 1967). Such procedures can add between 10 and 36000 MJ(t) to the total energy requirement of reprocessing one tonne of fissile-bearing material (see table K.2).

Intermediate-level wastes are raffinates from the second and third stages of reprocessing, amounting to about a thousand litres of liquid per tonne of fuel treated. Since such wastes contain fewer fission products than high-level wastes, they can either be discharged into the sea after a short period of storage or stored in tanks with de-cladding wastes (Blomeke, 1967). Neither method consumes as much energy as the disposal of high-level wastes, as table K.2 indicates.

Liquid low-level wastes from reprocessing can amount to one hundred thousand litres for each tonne of fuel treated. These contain only small amounts of radioactive material and such effluents are generally discharged directly into the environment, whilst low-level solids are usually buried in unlined pits (Blomeke, 1967). As indicated in table K.2 the energy input of this type of disposal is negligible in comparison with other waste management operations.

Table K.1 : Estimated energy inputs to the preparation, solidification and disposal of high-level liquid wastes.

Operation		Energy input (MJ/te fissile-bearing material) minimum	maximum
Concentration	:	14400(e)	14400(e)
Interim storage	:	3610(e) + 10(t)	36000(e) + 60(t)
Solidification	:	200(e) + 1600(t)	7200(t)
Containerisation	:	1200(e) + 2400(t)	2400(e) + 4800(t)
Transport	:		2000(e)
TOTAL (rounded)	:	19000(e) + 4000(t)	55000(e) + 12000(t)

Table K.2 : Estimated energy requirement of nuclear waste management.

Type of waste	Energy requirement (MJ/te fissile-bearing material) minimum	maximum
High-level :	19010(e) + 4000(t)	55000(e) + 48000(t)
Intermediate-level :	40(t)	4000(t)
Low-level :	5(t)	300(t)
TOTAL (rounded) :	19000(e) + 4000(t)	55000(e) + 52000(t)

Appendix L : Nuclear power plant construction

A nuclear power plant, or nuclear power station, is a collection of machinery that extracts energy, and generates saleable fuel, from nuclear material such as uranium. The power plant basically consists of a reactor, cooling system and turbo-generating machinery.

The reactor provides the conditions for heat-releasing fission reactions between neutrons and fissile nuclei such as uranium-235 (U-235) in fuel elements. Fundamental components of the reactor include a core of such fuel elements, moderator material that adjusts the speed of fission-producing neutrons and control rods that regulate the rate of fission reactions. These components are contained in a pressure vessel or in pressure tubes surrounded by suitable shielding material.

Heat is extracted from the reactor by the cooling system which consists of a series of pipework and heat transfer equipment. The primary circuit of the system which passes through the reactor itself contains a liquid or gaseous coolant that may, in certain cases, also act as a moderator. Secondary cooling circuits, which are connected to the primary circuit by heat exchangers, supply steam to the turbo-generating machinery.

Turbo-generating machinery enables heat energy obtained from the reactor to be converted first into mechanical energy and then into electrical energy. Steam from the secondary cooling circuits rotates turbines which drive electro-magnetic generators. This section of the power

station also contains other electrical equipment which governs the supply of electricity to the transmission and distribution network.

Six different types of burner reactor used in nuclear power stations are considered here; the MAGNOX reactor, the Advanced Gas-cooled Reactor (AGR), the Steam Generating Heavy Water Reactor (SGHWR), the CANDU reactor, the Boiling Water Reactor (BWR) and the Pressurised Water Reactor (PWR). As regards the basic constructional differences between these designs, the MAGNOX reactor uses a solid graphite moderator, carbon dioxide primary coolant and a stainless steel pressure vessel. The AGR also relies on a graphite moderator and carbon dioxide gas coolant but has a steel reinforced concrete pressure vessel. The SGHWR and CANDU designs both incorporate deuterium oxide, or heavy water, as a moderator and zirconium pressure tubes, rather than pressure vessels, for containing the fuel elements. The former reactor, however, has ordinary, or light, water for coolant, whilst the latter combines the deuterium oxide moderator as a coolant. The BWR and PWR systems are types of Light Water Reactor (LWR) which use ordinary water as coolant and moderator in a core enclosed by a stainless steel pressure vessel.

Many other materials are used in the nuclear power plant. Cement, sand, gravel and reinforcing steel are included in the construction of power station buildings. Mild and stainless steels, copper, aluminium and other metals are incorporated into cooling systems, turbine machinery

and electrical equipment. The type and quantity of such materials contained in the six main reactor designs was deduced by examining textbooks (Central Electricity Generating Board, 1971; Hunt, 1974), reactor manuals (International Atomic Energy Agency, 1967, 1970a, 1972; Nuclear Engineering International, 1972; Central Electricity Generating Board, 1973, 1975), technical reports (Gabriel and Smith, 1970; Moore, Hicks, Bradley and Rowlands, 1973; McKeague, 1974), industrial articles (Steigelmann, 1969/70; Canadian Nuclear Agency, 1973, 1974; Budwani, 1974; Fishlock, 1974; Russel, 1974) and economic assessments (Nuclear Power Advisory Board, 1974; Organisation for Economic Co-operation and Development, 1974; Nuclear Energy Agency, 1975).

This information was standardised for typical power stations with a net electrical output power rating of a thousand mega-watts "sent out" (1000 MW[e] - so). Estimates were then combined with results from the data base (see chapter 3) to deduce the total amount of energy required to build a nuclear power plant. Results for the six main burner reactor types are given in tables L.1, L.2, L.3, L.4 and L.5. These results include estimates of the energy used in the manufacture and transportation of all important materials as well as the fuel consumed by assembling components on the site of construction.

Table L.1 : Estimated energy inputs to the construction of a typical commercial MAGNOX power station.

Input (per 1000MW[e] - so)	Energy contribution (10^6MJ/1000MW[e] - so) minimum	maximum
Mild steel		
96 x 10^3 - 130 x 10^3 te	: 155(e) + 2550(t)	195(e) + 4015(t)
Stainless steel		
12 x 10^3 - 16 x 10^3 te	: 190(e) + 370(t)	315(e) + 640(t)
Concrete		
280 x 10^3 - 330 x 10^3 te	: 20(e) + 270(t)	25(e) + 410(t)
Graphite moderator		
11 x 10^3 te	: 165(e) + 635(t)	475(e) + 1220(t)
Construction power		
47 x 10^6 - 62 x 10^6kWhe	: 170(e)	225(e)
Construction fuel		
54 x 10^6 - 77 x 10^6 l	: 2340(t)	3305(t)
Miscellaneous metals		
and other items	: 155(e) + 3730(t)	200(e) + 3830(t)
TOTAL	= 855(e) + 9895(t)	1435(e) + 13420(t)

Table L.2 : Estimated energy inputs to the construction of a typical commercial AGR power station.

Input (per 1000MW[e]-so)	Energy contribution (10^6MJ/1000MW[e]- so) minimum	maximum
Mild steel		
74×10^3 - 110×10^3 te	: 120(e) + 2195(t)	145(e) + 3110(t)
Stainless steel		
11×10^3 - 13×10^3 te	: 150(e) + 290(t)	240(e) + 475(t)
Concrete		
280×10^3 - 300×10^3 te	: 20(e) + 240(t)	25(e) + 380(t)
Graphite moderator		
3×10^3 te	: 50(e) + 185(t)	140(e) + 350(t)
Construction power		
39×10^6 - 49×10^6 kWhe	: 140(e)	175(e)
Construction fuel		
47×10^6 - 60×10^6 l	: 2020(t)	2595(t)
Miscellaneous metals and other items	: 115(e) + 2165(t)	145(e) + 2740(t)
TOTAL	= 595(e) + 7095(t)	870(e) + 9650(t)

Table L.3 : Estimated energy inputs to the construction of a typical commercial SGHWR power station.

Input (per 1000MW[e]- so)	Energy contribution (10^6MJ/1000MW[e]- so) minimum	maximum
Mild steel		
85×10^3 - 110×10^3 te	: 120(e) + 2220(t)	155(e) + 3550(t)
Stainless steel		
12×10^3 - 18×10^3 te	: 265(e) + 355(t)	270(e) + 725(t)
Concrete		
215×10^3 - 250×10^3 te	: 20(e) + 250(t)	20(e) + 300(t)
Deuterium oxide moderator		
320 te	: 510(e) + 6840(t)	880(e) + 10800(t)
Construction power		
35×10^6 - 49×10^6 kWhe	: 125(e)	175(e)
Construction fuel		
41×10^6 - 56×10^6 l	: 1775(t)	2400(t)
Miscellaneous metals and other items	: 95(e) + 1830(t)	130(e) + 2460(t)
TOTAL	= 1135(e) + 13270(t)	1630(e) + 20235(t)

Table L.4 : Estimated energy inputs to the construction of a typical commercial CANDU power station.

Input (per 1000 MW(e) - so)	Energy contribution (10^6MJ/1000MW(e) - so)	
	minimum	maximum
Mild steel		
70×10^3 - 97×10^3 te	: 125(e) + 1940(t)	150(e) + 2955(t)
Stainless steel		
13×10^3 - 23×10^3 te	: 190(e) + 370(t)	490(e) + 965(t)
Concrete		
355×10^3 - 410×10^3 te	: 20(e) + 310(t)	35(e) + 470(t)
Deuterium oxide moderator		
970 te	: 1535(e) + 20740(t)	2675(e) + 32730(t)
Construction power		
42×10^6 - 58×10^6kWhe	: 150(e)	210(e)
Construction fuel		
49×10^6 - 66×10^6 l	: 2125(t)	2835(t)
Miscellaneous metals		
and other items	: 90(e) + 1690(t)	125(e) + 2295(t)
TOTAL	= 2110(e) + 27175(t)	3685(e) + 42250(t)

Table L.5 : Estimated energy inputs to the construction of typical commercial BWR and PWR power stations.

Input (per 1000MW[e]- so)	Energy contribution (10^6MJ/1000MW[e]- so) minimum	maximum
Mild steel		
53×10^3 - 55×10^3 te	: 70(e) + 1360(t)	105(e) + 2325(t)
Stainless steel		
25×10^3 - 31×10^3 te	: 325(e) + 660(t)	680(e) + 1355(t)
Concrete		
175×10^3 - 265×10^3 te	: 10(e) + 150(t)	30(e) + 355(t)
Construction power		
33×10^6 - 47×10^6 kWhe	: 120(e)	170(e)
Construction fuel		
30×10^6 - 50×10^6 l	: 1280(t)	2160(t)
Miscellaneous metals and other items	: 65(e) + 1265(t)	95(e) + 1955(t)
TOTAL	= 590(e) + 4715(t)	1080(e) + 8150(t)

Appendix M : The electricity supply network

All large scale commercial systems which generate electricity require some form of supply network to deliver the fuel from the site of production to the point of use. Most national electricity generation systems consist of a number of individual power stations interconnected by a transmission network, or 'grid'. This grid generally enables a steady flow of electricity to be delivered to specific distribution points despite fluctuations in the balance between local supply and demand. The grid is linked to consumers by means of a distribution, or 'mains', network.

In physical terms the supply network consists of many kilometres of electrical cables and wiring. The grid is usually comprised of overhead copper cables suspended from steel tower pylons or of underground copper and aluminium cables carried in specially designed pipes. Transformer equipment is installed at particular points to effect changes in voltage. The mains are composed of more copper cabling carried overground by small steel towers and timber poles or laid beneath the ground in distribution channels. More transformers are used to connect individual users or groups of consumers to the electrical circuit. In addition to these basic components, ancillary buidings, equipment and machinery are required to facilitate the control and maintenance of the system.

The provision of an electricity supply network can influence the energy analysis of electricity generation and delivery in three different ways. The amount of energy required to construct the system introduces a

preliminary, or 'capital', energy contribution and the energy used in all control, maintenance and repair work creates a concurrent, or 'operating', energy contribution. Additionally the efficiency of the transmission and distribution network determines the total amount of electricity actually available from the system, since losses due to resistance heating, current leakage, etc., cause the quantity of fuel delivered to consumers to be less than the amount initially supplied to the grid by power stations. The purpose of this appendix is to estimate the magnitude of these energy contributions and electrical network inefficiencies for the supply system of a typically industrialised country such as the United Kingdom.

The preliminary energy contribution can be sub-divided into the energy requirement terms describing the construction of; the grid, the mains, the link between the power station and the grid, and miscellaneous buildings and equipment. Energy inputs to the first three items were deduced by assessing the amount of energy required to construct standard lengths of typical grid and mains line. Estimates obtained by combining results of the data base (see chapter 3) with basic information on the use of materials and fuel in construction are shown in table M.1 and M.2. The amount of energy required to build sufficient transmission and distribution network for a typical one thousand mega-watt output capacity (1000MW(e)- so) power station was calculated by multiplying these unit energy requirements by the appropriate ratios of line length to power station

capacity (i.e. circuit-metres per 1000MW[e] - so) and lifetime estimates illustrated in table M.3. These factors were obtained from statistical information (Central Electricity Generating Board Statistical Yearbooks, 1952 - 1976; Handbook of Electricity Supply Statistics, 1976). Subsequent estimates of the total preliminary energy contribution, which include the energy requirements of constructing miscellaneous buildings and equipment deduced from financial data (Report on the Census of Production, 1968; The Electricity Council Annual Report, 1974 - 75), are presented in table M.4.

Estimates of the annual concurrent energy contribution of operating and repairing the system were obtained by combining basic information (Report on the Census of Production, 1968; The Electricity Council Annual Report, 1974 - 75) with figures from the data base (see chapter 3). Results are shown in table M.5. The ratio of the electricity delivered by the system to that originally supplied to it from power stations was evaluated from statistical information (Organisation for Economic Co-operation and Development, 1971; Report on the Census of Production, 1968; Department of Energy, 1975). This ratio was found to be;

$$\frac{\text{electricity delivered to consumers}}{\text{electricity produced by power plant}} = 0.90 \pm 0.05$$

Table M.1 : Estimated energy requirements for constructing a typical section of transmission grid.

Item	Input (per circuit-m)	Energy requirement (MJ/circuit-m) minimum	maximum
Copper	15.0 kg	117(e) + 532(t)	171(e) + 757(t)
Steel	6.7 kg	16(e) + 167(t)	18(e) + 326(t)
Cement	19.0 kg	7(e) + 116(t)	8(e) + 124(t)
Insulation	0.5 kg	1(e) + 7(t)	2(e) + 22(t)
Transformers	7×10^{-5} units	2(t)	2(t)
Misc. items	-	7(e) + 41(t)	30(e) + 185(t)
TOTAL	=	148(e) + 865(t)	229(e) + 1416(t)

Table M.2 : Estimated energy requirements for constructing a typical section of distribution mains.

Item	Input (per circuit-m)	Energy requirement (MJ/circuit-m) minimum	maximum
Copper	0.78 kg	6(e) + 28(t)	9(e) + 39(t)
Cement	16.0 kg	6(e) + 97(t)	7(e) + 104(t)
Transformers	2×10^{-3} units	2(e) + 38(t)	3(e) + 51(t)
Steel	0.21 kg	10(t)	1(e) + 25(t)
Insulation	0.05 kg		
Timber	3×10^{-3} m^3		
Misc. items	-	9(t)	3(e) + 33(t)
TOTAL	=	14(e) + 182(t)	27(e) + 252(t)

Table M.3 : Estimated factors for UK transmission grid and distribution lines.

Network	Ratio of line length to unit capacity (circuit-metres/1000MW(e)-so)	Ratio of power plant life to line life*
Grid :	$3 \times 10^5 - 5 \times 10^5$	1 - 2
Mains :	$5 \times 10^6 - 1 \times 10^7$	0.5 - 1
Power plant to grid :	2×10^4	1

* assuming an average power station life of 30 years.

Table M.4 : Preliminary energy contribution of the electricity supply network.

Item		Energy input (10^6 MJ/1000MW(e)-so) minimum	maximum
Grid network	:	45(e) + 260(t)	230(e) + 1415(t)
Mains network	:	35(e) + 455(t)	230(e) + 2520(t)
Network from power plant to grid	:	15(e) + 220(t)	35(e) + 500(t)
Miscellaneous buildings and equipment	:	30(e) + 765(t)	40(e) + 1095(t)
TOTAL	=	125(e) + 1700(t)	535(e) + 5530(t)

Table M.5 : Concurrent energy contribution of the electricity supply network.

Item	Energy input (10^6 MJ/1000MW(e)-so/year) minimum	maximum
Maintenance and repair of the network :	50(t)	5(e) + 60(t)
Maintenance and repair of ancillary equipment and buildings :	20(t)	25(t)
Operation of miscellaneous plant and vehicles :	50(t)	55(t)
TOTAL =	120(t)	5(e) + 140(t)

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